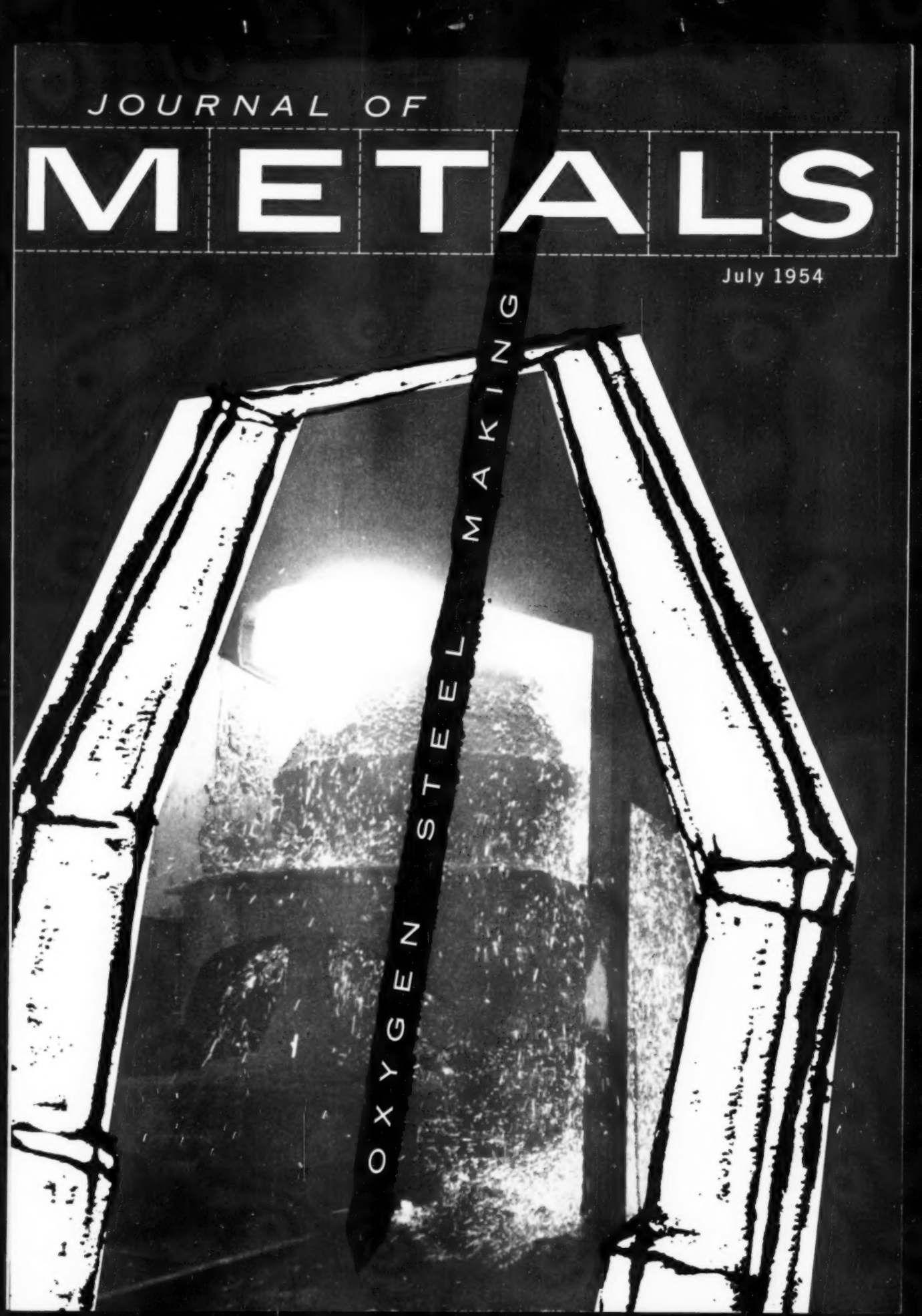
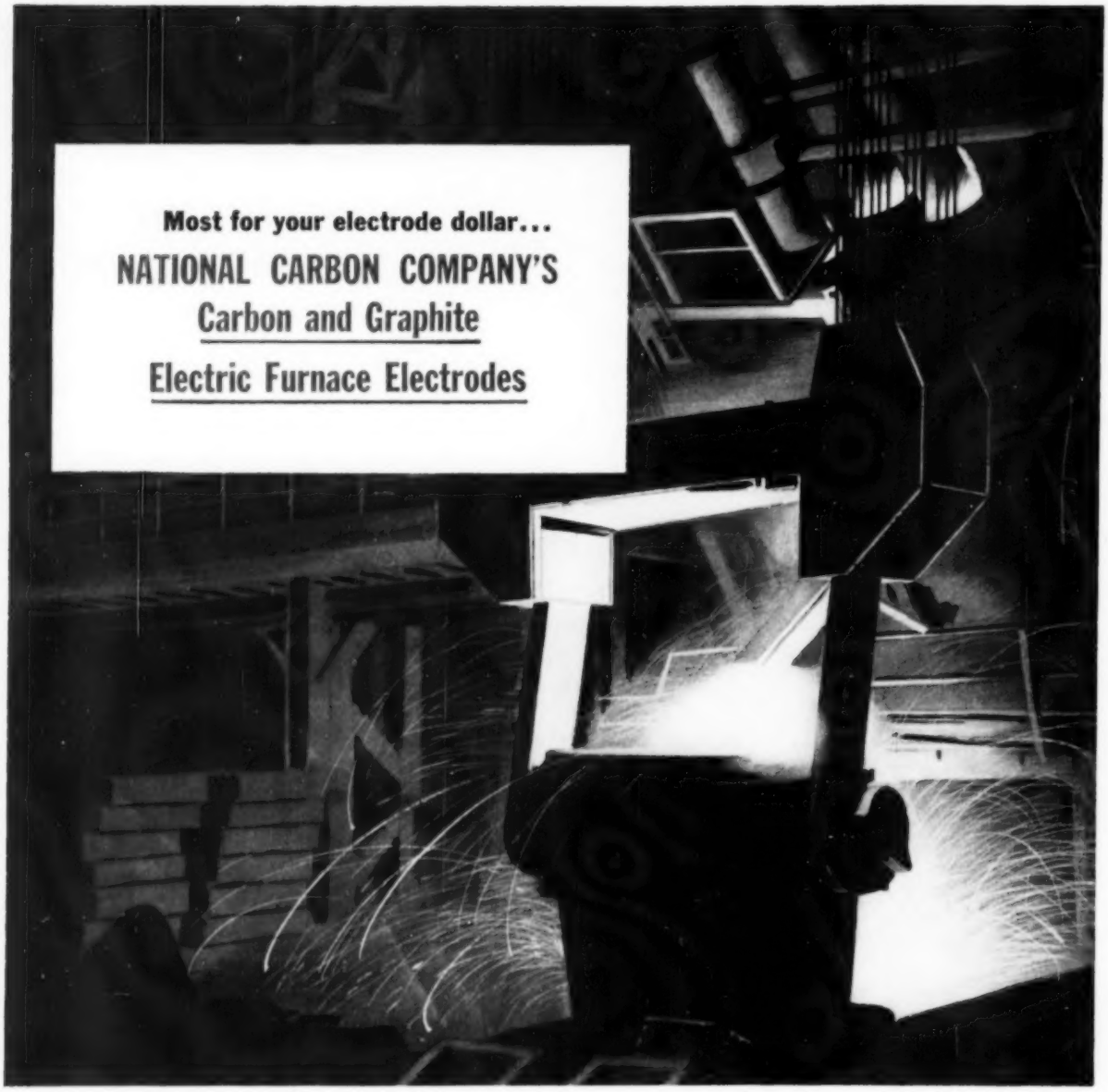
JOURNAL OF

METALS

July 1954





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VOL. 6, NO. 7 JULY, 1954

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COVER

Oxygen steelmaking as developed in postwar Europe has captured the imagination of steelmen in the U. S. Harry Brocke illustrates this process on the cover, and three articles dealing with the subject appear on pp. 804, 817, and 829.

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Meet The Authors

Otwin Cuscoleca (p. 817) has been associated with the Oesterreichisch-Alpine Montangesellschaft, (OAMG), since the end of World War II. He is a managing technical director for the company. Mr. Cuscoleca has taken an active part in reorganizing the ore mines and steel plants of the firm. Previously, he had been employed by the August-Thyssen-Hutte-AG, Duisburg, at one time Germany's largest steel plant. Mr. Cuscoleca has authored several papers and lectured at international metallurgical meetings on oxygen steelmaking. In 1949 he was in the U. S. visiting steel plants. He holds membership in AIME and other metallurgical societies of Western Europe.

O. CUSCOLECA

W. R. OPIE

W. R. Opie (p. 807) is supervisor, geochemical and metallurgical research, research laboratory, titanium div., National Lead Co., South Amboy, N. J. Dr. Opie was born at Butte, Mont., and graduated from the Montana School of Mines (B.S.) and Massachusetts Institute of Technology (Sc.D.). For approximately three years he was a foundry metallurgist with Wright Aeronautical. After one year in the U. S. Navy, Dr. Opie joined the staff at Anaconda Copper Corp., as a design metallurgical engineer. From 1946 to 1948 he was a research assistant at MIT. Dr. Opie then accepted the position of research metallurgist with the American Smelting & Refining Co. In 1950 he joined the National Lead Co. An AIME Member, he counts golf, fishing, and gardening among his hobbies. *Solubility of Hydrogen in Aluminum* and *Solubility of Hydrogen in Lead* are two articles he has co-authored.

R. C. Buehl (p. 814) joined the Bureau of Mines in 1943 as chief, ferrous pyrometallurgy section, Pittsburgh, a position he presently holds. Born in Brooklyn, Dr. Buehl is a graduate of Polytechnic Institute and Massachusetts Institute of Technology. From 1939 to 1942 he had been an industrial fellow at Mellon Institute. He holds membership in AIME and the Science of Metals Club. Previous articles of his that have appeared are: *Manganese for the Steel Industry* and *Manganese*

R. C. BUEHL

J. T. RICHARDS

from Open Hearth Slag Could Supply One Half of U. S. Needs. His hobbies take him out of doors: Bird watching, skating, and skiing.

John T. Richards (p. 827) graduated from Cornell University in 1943 (B.S., administration engineering). Mr. Richards joined the Beryllium Corp. in 1946 as development engineer. He remained with this firm until 1953 when he became president and chief engineer of Penn Precision Products, Inc., Reading, Pa. Mr. Richards is a member of AIME, ASME, ASTM, SAE, ASM, and the British Institute of Metals. He has written articles on beryllium that have appeared in JOURNAL OF METALS in May 1951 and May 1954.

H. B. Emerick (p. 804) attended Carnegie Institute of Technology. His entire business career has been with the Jones & Laughlin Steel Corp. He joined the Aliquippa Works div. in 1935 and rose to assistant chief metallurgist. He was transferred to the general offices in Pittsburgh as assistant to the vice-president, technology, in 1951. Mr. Emerick was appointed assistant director of technical services, his present position, in April 1953. He has authored several articles for the National Open Hearth Conference and in 1942 was co-author of the McKune Award paper, *Duplex Process for the Manufacture of Open Hearth Steel.* He also wrote *Survey of Factors Affecting Surface Quality of Semi-Finished Steel.* He was also on the editorial board for the second edition of *Basic Open Hearth Steelmaking.* With all these activities, Mr. Emerick still finds time to enjoy photography and power boating. He

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Edwin M. Meyer (p. 811) born in Chicago, Ill., is presently residing in Cranford, N. J. Mr. Meyer received his degrees from the Armour Institute of Technology (B.S.) and Massachusetts Institute of Technology (S.M.). Following graduation, he joined the G & W Electric Specialty Co., as an electrical engineer. In 1930 he accepted a similar position with Porcelain Products, Inc. From 1939 to 1944 Mr. Meyer was chief engineer for Victor Insulators, Inc., Victor, N. Y. He was associated with the Delco appliance div. of General Motors Corp. for six years as plant engineer. Since 1950 Mr. Meyer has been chief engineer for the U. S. Metals Refining Co., Carteret, N. J. Mr. Meyer is a member of AIEE and is a licensed professional engineer in New York and New Jersey. He admits to being an amateur in his hobbies: Golf and the electric organ.

Pierre Coheur (p. 829) is director, Centre National Recherches Metallurgique, Liege, Belgium. Dr. Coheur was born at Herstal, Belgium and graduated from the University of Liege (1937, 1942). He has been a professor on the staff at the University. Dr. Coheur holds membership in AIME, ASTM, ASM, Verein Deutscher Eisenhuttenleute, Societe Francaise de Metallurgie, Iron and Steel Institute (London), Societe Royale Belge des Ingenieurs, and other engineering societies in Europe. In his spare time, Dr. Coheur enjoys playing golf and skiing.

P. COHEUR

T. R. A. DAVEY

T. R. A. Davey (p. 838), born in Melbourne, Australia, is research officer for the Broken Hill Associated Smelters, Pty., Ltd. Mr. Davey is a graduate of the University of Melbourne and has been associated with Broken Hill since 1947. Music, literature, chess, table tennis, and tennis occupy this author in his free time. He also holds membership in the Institute of Mining and Metallurgy. In August 1953, he wrote *Vacuum Dezincing of Desilverized Lead Bullion.*

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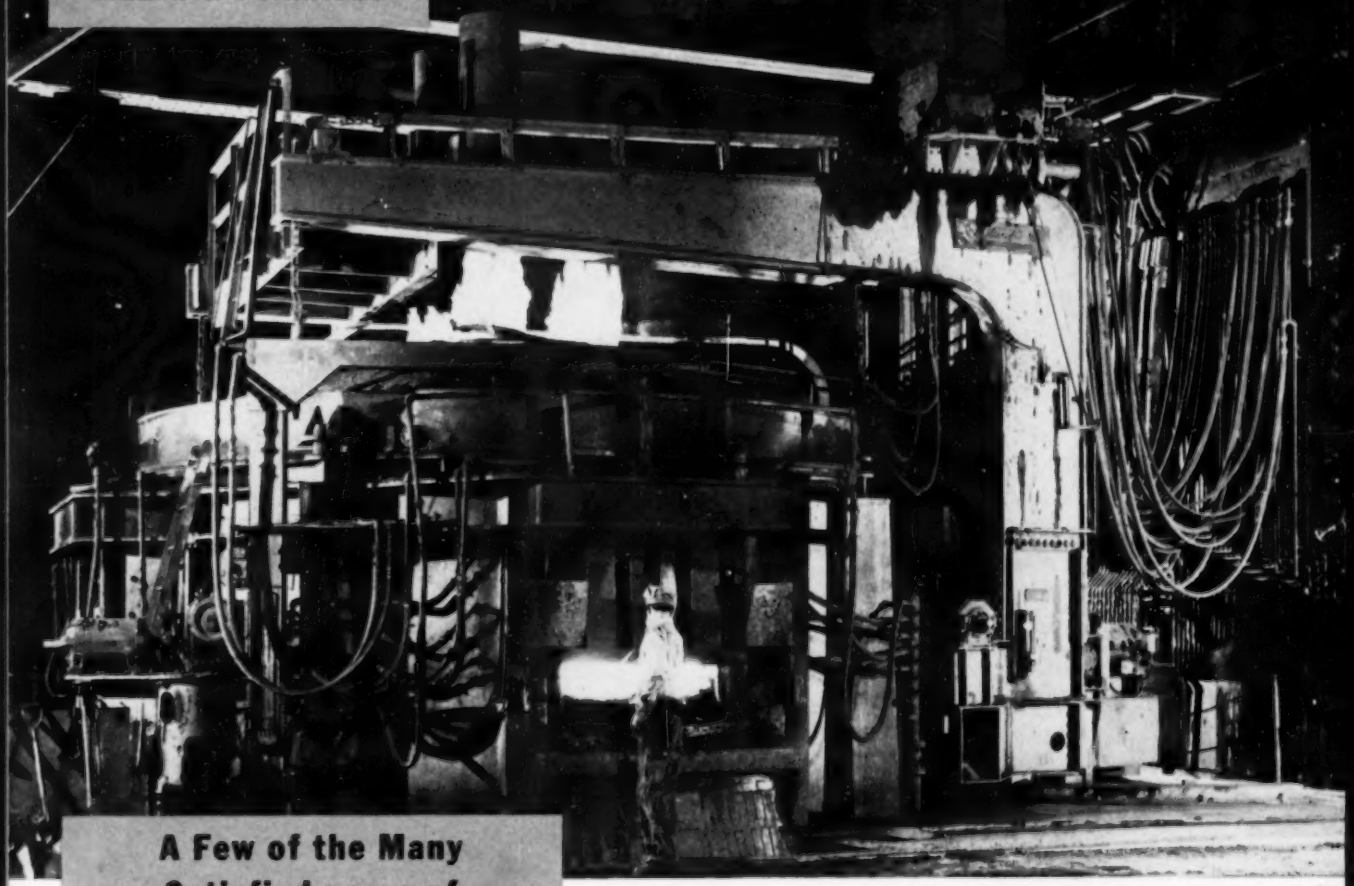

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Books for Engineers

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Titanium and Titanium Alloys, by John L. Everhart. *Reinhold Publishing Corp.* $3.00, 184 pp., 1954.—This book is aimed at the engineer or designer interested in the possibilities of applying titanium to the solution of his problems, and is not intended for the titanium producer. Only enough data is supplied to indicate the effects of melting methods on properties. Titanium properties are discussed briefly to furnish background for discussion of commercial materials.

Sulfur. System No. 9, Section A3 of the eighth edition of Gmelin's Handbook of Inorganic Chemistry. *Edited by the Gmelin Institute,* $34.00, 252 pp., 1953.—The present section A3 which concludes part A of the Gmelin sulphur volume is devoted to the physics and chemistry of elemental sulphur. It begins with a systematic survey of the numerous sulphur forms and clearly analyzes this topic. Purification of sulphur in the laboratory is discussed as well as the preparation of modifications and concentration and separation of isotopes. Material is arranged with the help of the phase diagrams, and contradictory data is clarified. The chapter on physics covers in the usual sequence the nucleus, the atom including particularly the atomic and ionic radii, the molecule with particular reference to the existence of the different molecule types in steam, as well as crystallographic, magnetic, and electrical properties.

The Microscopy of Metals. *The Institution of Metallurgists,* 132 pp., 1953.—The volume is derived from the Institution of Metallurgists refresher course given in the autumn of 1953 at London, England. Topics covered are: Microscopes and Microscopic Techniques, by G. K. T. Conn; General Metallography, by H. Evans; Hot Stage Microscopy, by P. J. E. Forsyth; and Phase-Contrast and Interference Metallography, by D. McClean.

Engineering Analysis, by D. W. Ver Planck and B. R. Teare, Jr. *John Wiley & Sons, Inc.,* $6.00, 344 pp., 1954.—The subject matter lays special stress on the thinking process involved in translating engineering situations into mathematical language. The subject is developed by example. Specifically, the book deals with such matters as: Defining the problem to be solved; deciding what principle to apply; choosing coordinate systems; checking thoroughly; choosing dimensionless variables; and the sketching of curves.

Metals and How to Weld Them, by T. B. Jefferson and Gorham Woods. *The James F. Lincoln Arc Welding Foundation,* $2.00, 322 pp., 1954.—The book was written for those who want a better understanding of what happens when metals are welded. The text has been designed for classroom and homestudy. Welding of various metals are discussed in separate chapters. Problems liable to be encountered in welding any commercial metal, from aluminum to tool and die steels, are covered completely. A special chapter is devoted to hard facing. How to make good welds is treated in another chapter.

Industrial Diamond Trade Names Index and Yearbook, fifth edition, compiled jointly by Industrial Diamond Information Bureau and Industrial Diamond Review. *N. A. G. Press Ltd.,* 124 pp., 1954.—The volume contains about 2500 trade names. A data sheet supplement gives information on a number of subjects of interest to users and producers of diamond tools and the physical and chemical properties of diamonds. In addition data is presented on the crystallography of diamonds and the care of diamond truing tools, shaping of diamond tool standards.

Sulphur Data Book, compiled by Freeport Sulphur Co. *McGraw Hill Book Co.,* $5.00, 140 pp., 1954.—According to the publishers, the book marks the first time that a comprehensive collection of data on sulphur has been presented under one cover. It includes material from literature, from private sources, and from experience gained by Freeport in mining and research work. Major sections deal with nature of sulphur, its physical and chemical properties, reaction thermodynamics, solubilities, and methods of analysis.

Industrial Electroplating, second edition, by E. A. Ollard and E. B. Smith. Published for Metal Industry by *Iliffe & Sons Ltd.,* 364 pp., 1954.—This second edition contains new sections dealing with water and drainage, the purification of solutions, safety precautions, and ventilation in plating shops. Many new formulas for solutions as well as details of the latest advances in testing deposits are given. Additional tables for reference are provided.

Boron Steels, Production and Use. *Organization for European Economic Cooperation,* $2.00, 142 pp., 1954.—The book is a result of Technical Assistance Mission 124 of the Office of European Economic Cooperation. The mission made a study of boron steels in Germany, Austria, Belgium, France, Italy, the Netherlands, and the United Kingdom. The book is the report on the technology of the countries visited.

Metals Properties, edited by Samuel L. Hoyt and sponsored by the Metals Engineering Handbook Board of the American Society of Mechanical Engineers. *McGraw-Hill Book Co.,* $11.00, 433 pp., 1954.—The book is a reference of specific information for the designer about properties of metals with which he works—properties such as strength, hardness, machinability, electrical conductivity, thermal conductivity, and others. The book brings data on more than 500 metals in common industrial use—AISI steels, ASTM steels, cast copper alloys, aluminum alloys, tin, magnesium, and many more.

Progress Reports of Investigation of Railroad Rails and Joint Bars, by Ralph E. Cramer and Russell S. Jensen. *University of Illinois.* 30¢, 40 pp., 1954.—The bulletin covers Investigation of Failures in Control-Cooled Railroad Rails, Twelfth Progress Report of the Rolling-Load Tests of Joint Bars, Twelfth Progress Report on Shelly Rail Studies at the University of Illinois, and Tests of Electric Flash Butt-Welded Rails.

Proceedings of a Conference on the Utilization of Scientific and Professional Manpower, prepared by Howard A. Meyerhoff. *Columbia University Press,* $3.50, 197 pp., 1954.—The results of the Arden House Conference sponsored by the National Manpower Council held Oct. 11 to 17, 1953 are covered. Major problems of utilization are introduced by J. D. Zellerbach and are elaborated on in papers by K. E. Boulding, Frank Pace, Jr., and S. L. Wolfbein.

Book of States. *Council of State Governments,* 1313 E. 60th St., Chicago 37. $10.00, 686 pp., 1954.—This new edition presents information on the 48 state governments, including constitutional developments, taxation and finance, administrative systems, and rosters of state officials.

Fundamentals of the Working of Metals, reviewed in the April 1954 issue of JOURNAL OF METALS, is available in the U. S. through Interscience Publishers, Inc.

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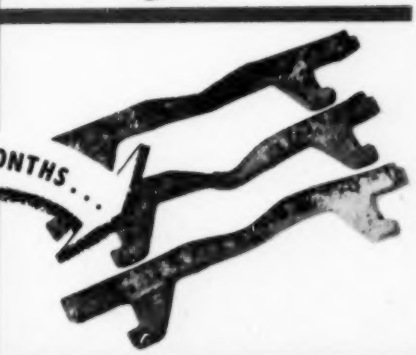

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Metallurgical Engineer, metallurgical graduate, with some experience in physical metallurgy, material specifications failure investigations, metallography or material testing. Location, Pennsylvania. Y9991(b).

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Metallurgical Engineer, under 30, with training and experience in crushing, grinding and other process operations in mineral dressing. Salary open. Location, Midwest. D9374.

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Metallurgical Engineer, recent graduate, to assist in modernization program of large eastern nonferrous refinery. Salary open. Location, New Jersey. Y9630.

Mill Metallurgist to take full charge of all testing and research. Graduate, experienced in operation of milling plants using crushing, grinding and froth flotation, etc. Salary to start $9600 a year. Location, Chile, S.A. F9578(a).

Research Physical Metallurgist, B.S. Must have two years experience in research and development, and have knowledge of research and development in physical metallurgy. Must be U.S. citizen. Salary, $6000 to $7200 a year. Employer will negotiate placement fee. Location, Chicago. C1787(b).

Metallurgist with Ph.D. in metallurgy, 25 to 40, experience not necessary, for company engaged in the production of uranium fuel elements beginning with concentrates and ending with the finished shape of fuel element. Salary open. Location, Ohio. Y9853.

Junior Metallurgist with nonferrous experience for research in copper powder, electrodeposition, atomization, reduction. Salary, $4420 to $5460 a year. Location, New Jersey. Y9849.

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Metallurgist, B.S. in mining engineering. Ten years experience all phases simple copper smelting, having had both technical and practical experience in all phases. Desires position research relative to extractive metallurgy. M-127.

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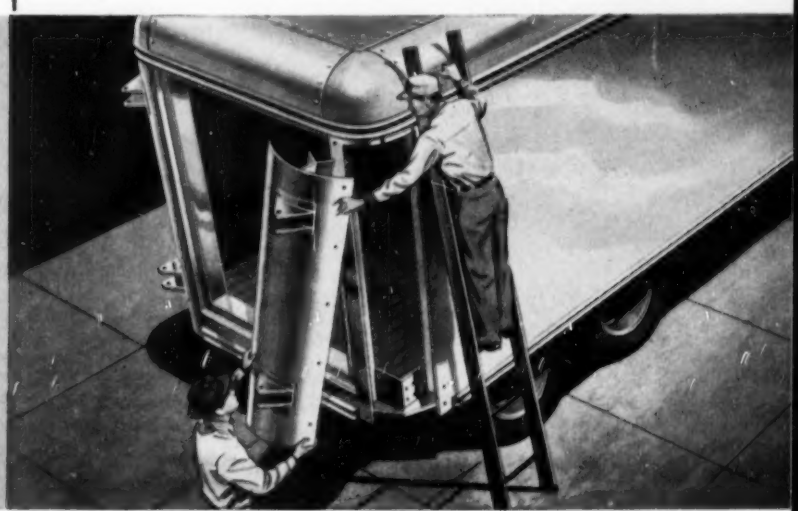

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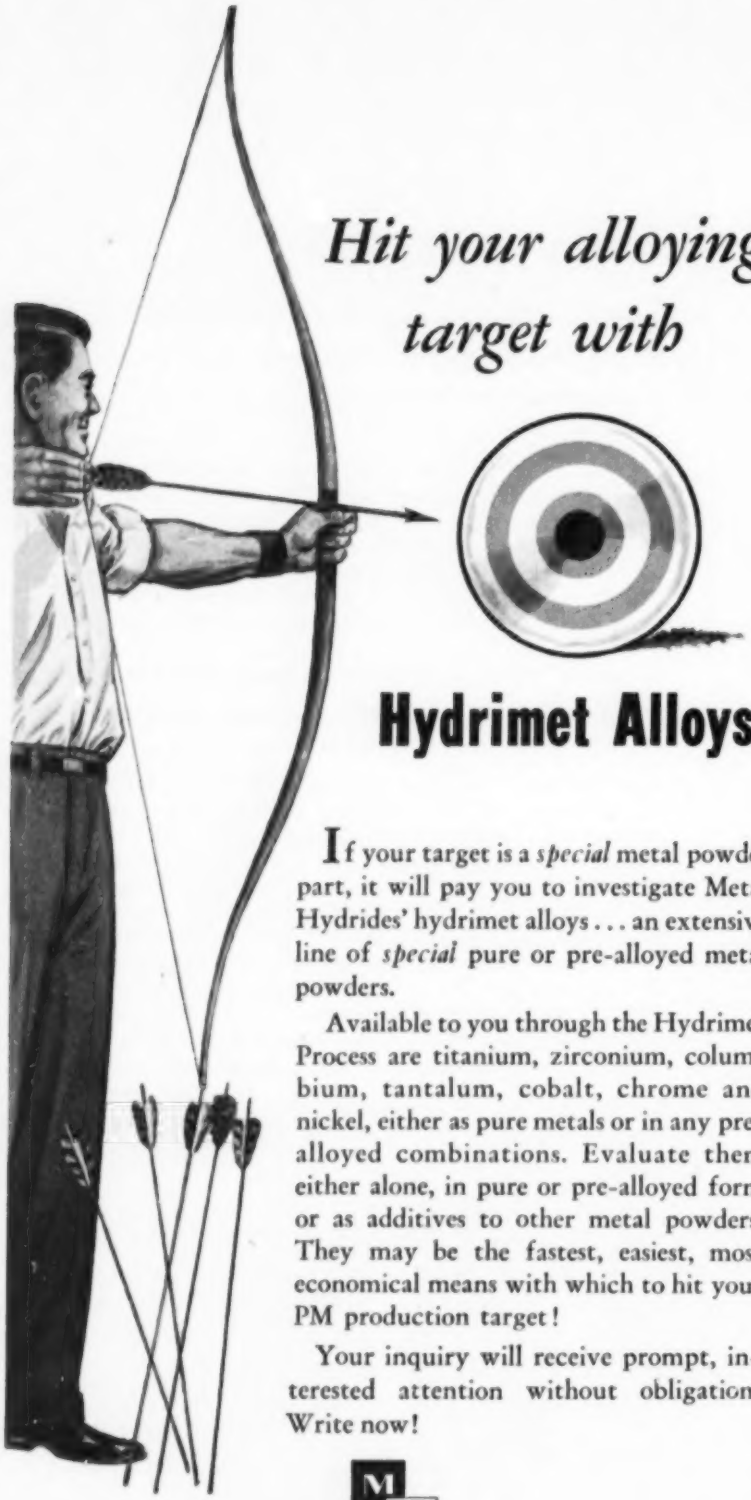

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. . Industrial Notes . .

• F. J. Stokes Machine Co. launched a $1 million expansion program designed to enlarge production facilities by 40 pct. Ground is to be broken for a 50,000 sq ft addition to the Stokes plant at 5500 Tabor Road, Northeast Philadelphia.

• **Henry F. Dever, president of Brown Instruments div., Minneapolis-Honeywell Regulator Co., in a paper before the National Federation of Financial Analysts Societies, said that automation is neither too difficult nor too costly for corporate management consideration. "Much in the way of improvement is available now, but more will be certainly available as the many kinds of equipment needed for automation are developed and perfected."**

• The world's largest tapered sheet and plate rolling mill went into production at the Davenport, Iowa Works of Aluminum Co. of America. Tapered aluminum sheet and plate for military aircraft is coming off the 144-in. mill in sizes up to 10 ft wide.

• National Tube div., U. S. Steel Corp., is installing equipment for manufacturing plastic pipe. The installation will be located at the Gary Works with actual pipe extrusion scheduled for September. Screw-type extruders will turn out ½ to 6 in. pipe using several plastic materials including polyethylene and polyvinyl chloride.

• *Inland Steel Co. has placed in operation a second continuous galvanizing line and retired the last of its outmoded molten zinc pots. Both of the firm's continuous lines will turn out Ti-Co brand galvanized sheets under the Sendzimir patent process. The second line represents an addition of about 33 pct in galvanized sheet capacity and means possible production of approximately 200,000 tons per year.*

• **Wilputte coke oven div., Allied Chemical & Dye Corp., has been awarded a contract for design and construction of 76 Wilputte underjet byproduct coke ovens with auxiliary equipment for Youngstown Sheet & Tube Co.'s Campbell Works, Youngstown, Ohio.**

• *Rust Furnace Co. is scheduled to construct two heat treating furnaces for pipe annealing at the Aliquippa Works of Jones & Laughlin Steel Corp. New in design, the furnaces, of the continuous walking beam type, will serve the new seamless tube specialties dept. They will be 54 ft 9 in. wide x 45½ ft long, and 6 ft high in the heating zone and 4 ft high in the soaking zone. Coke oven gas will be used as fuel with natural gas as a standby.*

New Products

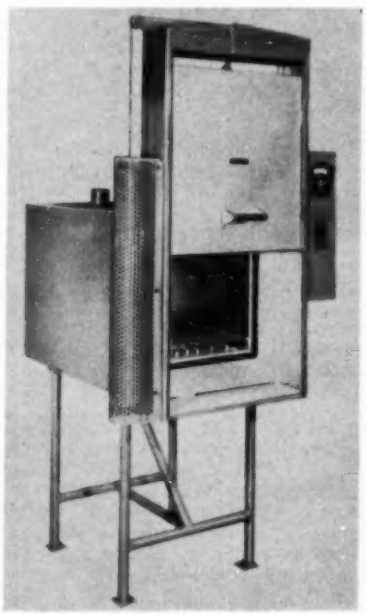
New Products | New Literature | New Services

For Further Information or Literature on any Product, Fill in the Coupon and Send to JOURNAL OF METALS
Students are Requested to Apply Directly to Manufacturers for Bulletins and Information

1—INVERTER: Development of a small nonresonant reed-type inverter capable of operation over wide frequency and ambient temperature ranges has been announced by the *Bristol Co.* This inverter is capable of converting low power dc signals to alternating voltages.

2—ROTARY TABLE: Successful application of a rotary table for surface grinders is announced by *Vulcan Tool Co.* It uses regular air line sources for power and attains variable speeds of 40 to 100 rpm. The table is a portable attachment for the surface grinding of flanged studs or bushings, bearing spacers, forming rolls, (angle or radius), convex or concave surfaces, flush pin gages, punches or dies (radius or angles).

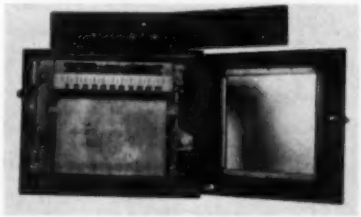
3—ELECTRIC OVEN: Model 100, type 1, offered by *Associated Metal Fabricators* features temperature



range, 200° to 900°F. Oven space is 15x15x36 in. deep. It has a spring loaded counterweighted door and 6 in. magnesia insulation.

4—RUST PREVENTION: Rust prevention made easy with push button dispensers is available from *E. F. Houghton & Co.* It is handy for protecting small items such as tools, dies, gages, and precision parts.

5—STRAIN SCANNING: Records from SE-4 strain gages at 50 points can be taken continuously and auto-



matically in a total of 30 to 90 sec with new SR-4 strain scanning and recording instruments available from *Baldwin-Lima-Hamilton Corp.* The recorder provides three ranges of measurement: 0 to 2000, 0 to 5000, and 0 to 10,000 micro-in. and can handle gages with mixed resistance ranging from 60 to 500 ohms.

6—ADHESIVES: New structural adhesives for bonding metal to metal and metal to plastic have been developed by the *Rubber & Asbestos Corp.* One offers extremely high strengths at normal temperatures and another offers extreme flexibility at elevated temperatures.

7—POROSITY TEST: A wetting agent with adequate sudsing properties has been developed for testing castings for porosity. The agent is introduced into the water and fine pin holes and microscopic openings which do not show up readily with plain water are quickly spotted. The sudsing agent causes the solution to bubble profusely when used in water testing of castings. The material is used about 1 oz per gal of water. *Magnus Chemical Co.*

8—TEMPILSTIKS®: Temperature indicating crayons for 413° and 438°F are now regularly available from *Tempil® Corp.* These newly added items make it possible to determine temperature intervals intermediate to 400°, 425°, and 450°F.

9—SAFETY: A rubber tired ball bearing cart has been designed by *Ansul Chemical Co.*, to transport the firm's dry chemical fire extinguishers quickly to the scene of a fire. The Pull-it is 44 in. high, 16 in. wide, and 13 in. deep.

10—REAGENT: A new reagent for iron analysis is available from *Fisher Scientific Co.* The reagent indicates *redox* endpoints and gives a Beer-Lambert curve for traces of iron.

11—BR HARDNESS TESTER: Hardness tests can be made at the rate of 10 or more specimens per min with the new conveyorized tester developed by the *Tinius Olsen Testing Machine Co.* When positioned, a preset load, from 500 to 4000 kg is applied to the specimen automatically by the steel ball and held for the period set on the electric timer.

12—SLIDE RULE: An 8 in. diam circular slide rule of laminated plastic has been added to the slide rule line at *Allegheny Plastics, Inc.*

13—DO SOLENOID VALVES: A complete line of diaphragm operated solenoid valves is now produced by *Eclipse Fuel Engineering Co.* They are now available in ⅜, ½, ¾, 1, 1¼, and 1½ in. pipe sizes. These valves are used for control of air, water, oil, propane and butane fuel gases, and sulphur dioxide gas. They will operate on line pressures as high as 150 psi and as low as ½ in. of water.

14—CONVEYOR: A new type of reciprocating conveyor designed to feed a constant volume of any dry or moist material from an overhead bin is announced by the *Hydro-Blast Corp.* Operation is not affected by variations in the consistency of the material.

15—MAGNETIC-PARTICLE TEST: A test unit that weighs only 45 lb is obtainable from *North American Philips Co., Inc.* It is designed to locate surface defects in rough castings, bar stock, forgings, and shop welds. The unit has two heavy duty, oil resistant insulated cables with

heavy duty, renewable prods that carry currents up to 600 amp at a maximum voltage of 1.5 v.

16—CURRENT TRANSFORMER: A butyl molded miniature current transformer has been announced by *General Electric Co.* It is designed for operating meters and instruments and can be used on either single phase two wire or polyphase circuits.

Free Literature

20—TI TUBING: Properties, applications, and advantages of titanium tubing are presented in bulletin No. 42 offered by *Superior Tube Co.* Tube sizes of seamless and weldrawn titanium are listed as well as tolerances, chemical analysis, and finishes.

21—INDUCTION FURNACES: Low frequency, coreless, crucible type electric melting and holding induction furnaces for nonferrous and gray iron foundries are described on file sheet issued by *International Selling Corp.*

22—INSTRUMENTATION INDEX: An alphabetical index covering all issues of the Instrumentation Magazine published from 1943 through 1953 is available from *Minneapolis-Honeywell Regulator Co.* Booklet 200-C lists all volumes and numbers published, article index by subject, and companies and institutions covered.

23—DICYANIDIAMIDE: The physical and chemical properties of Aero dicyanidiamide and its applications are described in booklet published by *American Cyanamid Co.*, industrial chemicals div.

24—TEMPERING: Catalog TD2-625 gives operating results being obtained in plants using Homo furnaces. Photos of typical installations, cut-away views of the furnaces, and specifications are listed. *Leeds & Northrup Co.*

25—TERNALLOY: Complete engineering information on the alloy series' mechanical and physical properties are contained in file folder brochure issued by *Apex Smelting Co.* High machinability of Ternalloy, which approaches 5000 sfpm, is explained and compared with various other metals and alloys.

26—REFRACTORIES: Coleman XX and Coleman refractories are detailed in bulletin obtainable from *Harbison-Walker Refractories Co.* Charts show spalling resistance, results of load tests, and thermal expansion.

27—HEAT EXCHANGER TUBES: The complete story on aluminum heat exchanger tubes is presented in booklet issued by *Aluminum Co. of America.*

28—ELECTREAT: Advantages of this heat treated cold finished steel are given in booklet issued by *Jones & Laughlin Steel Corp.* Illustrations are included.

29—CARBIDE TOOLS: Catalog including design specifications on cutter bits, finger bits, auger drills, grinding wheels, and other accessories is announced by *General Electric Co.*, Carboloy dept.

30—RADIOGRAPHIC UNIT: Folder describing a new portable self-contained industrial X-ray radiographic unit is available from the *North American Philips Co., Inc.* The literature explains the X-ray head, control and other features of the unit.

31—FATIGUE MACHINE: Sonntag model SF-4 fatigue machine of 13,000 lb capacity is presented in two page bulletin issued by *Baldwin-Lima-Hamilton Corp.* Elevated temperature testing, principles of constant force operation, and specifications are included.

32—COLD ROLLED STEELS: Up-to-date information on cold rolled specialties is contained in booklet TM9 is announced by *Crucible Steel Co. of America.* Melting, condition or temper, edge, finish, and packing data of stainless, alloy, and carbon spring steels is given. The booklet contains many helpful photographs, diagrams, and charts.

33—BRONZE: Advantages of continuous cast bearing bronze are explained in bulletin published by *American Smelting & Refining Co.* The process by which a variety of bronze alloys is cast into rods, tubes, and shapes of different diameters and profiles is described in detail.

34—SPECTROCHEMICAL ANALYSIS: Features and specifications of this all-in-one excitation unit for precision spectrochemical analysis are covered in catalog V3-54 released by *Jarrell-Ash Co.*

35—GRINDING DISKS: Bulletin describing Manhattan Moldiscs for portable grinders and sanders has been issued by Manhattan rubber div., *Raybestos-Manhattan, Inc.*

36—WET BLASTING: Information on finishing forging dies by the wet blasting process is contained in bulletin 103 published by *American Wheelabrator & Equipment Corp.*

37—PLATE SIZES: Data on available plate sizes of carbon, alloy, and clad steel are contained in folder obtainable from *Lukens Steel Co.* Thicknesses and weights are also included.

38—CONVEYING: A booklet concerned with permanent nonelectric magnetic equipment designed for steel handling and conveying has been published by *Eriez Mfg. Co.*

39—LOW FREQUENCY FURNACES: Low frequency, coreless, crucible type electric melting and holding induction furnaces for nonferrous and gray iron foundries are listed on pamphlet available from *International Selling Corp.*

40—SPECTROGRAPHY: Catalog illustrates and describes in detail the E612 Hilger two prism glass Raman spectrograph. The Hilger Raman source FL-1, and control units, cameras, oscillators, amplifiers and other accessory equipment are also illustrated and described. *Jarrell-Ash Co.*

Lebanon Steel Foundry has published a 32 page book on the entire process of steel casting manufacture from the design and engineering phase to final applications. A brief history of Lebanon Steel and early development of circle L castings are mentioned. A copy may be obtained by writing on company letterhead to the Lebanon Steel Foundry Co., 107 Lehman St., Lebanon, Pa., attention: William H. Worrilow, Jr.

Journal of Metals **July**
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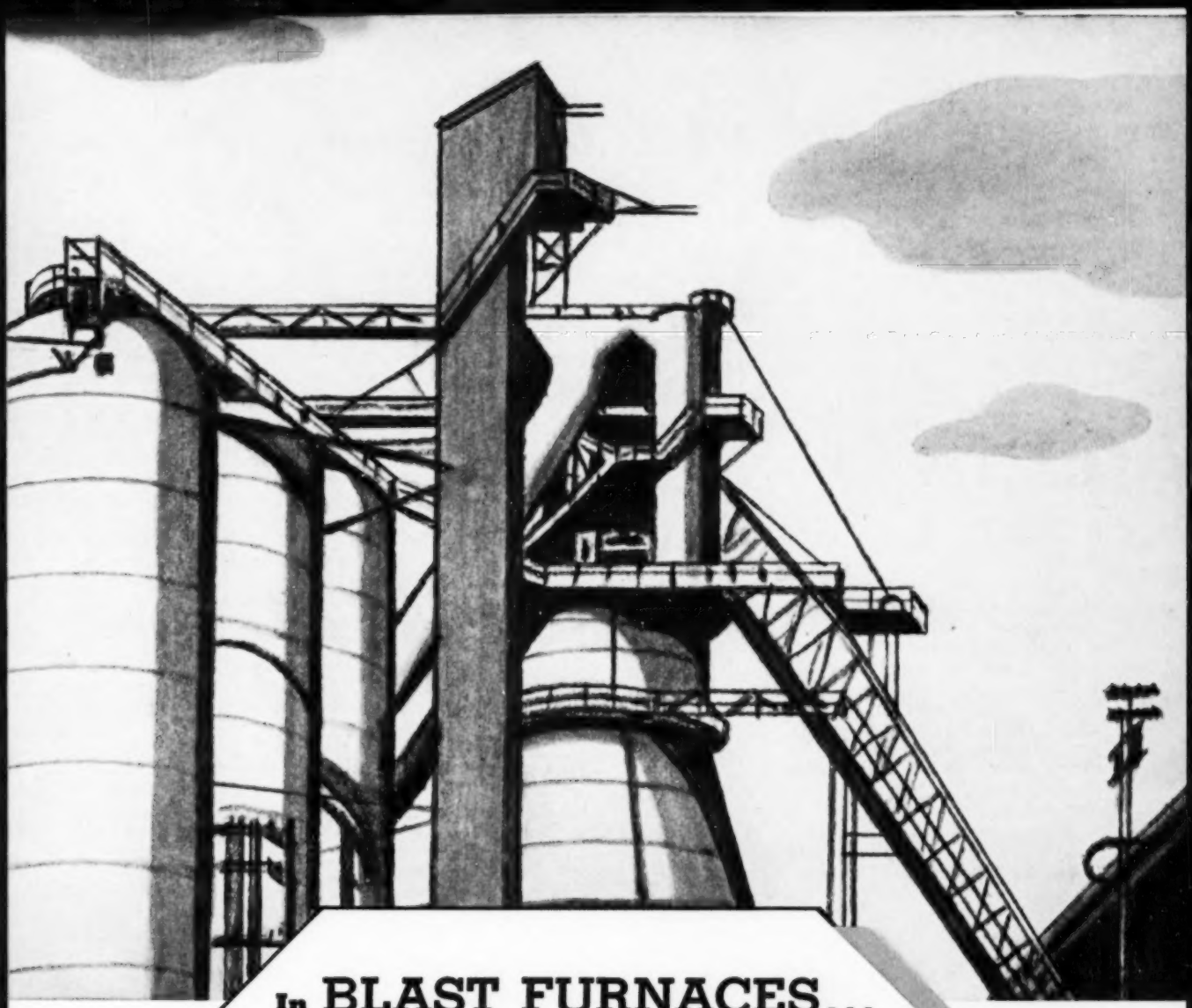

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Journal of Metals

OXYGEN steelmaking, the biggest thing to hit the European iron and steel industry in recent years, is being launched in the U. S. by Kaiser Engineers div., Henry J. Kaiser Co., who signed a licensing agreement with Brassert Oxygen Technik A. G., of Zurich, Switzerland.

Thus, as sole U. S. licensor of the European group, Kaiser Engineers join the world-wide list of member firms which have developed the new approach to steelmaking. Three plants are operating in Europe on a commercial basis and one plant is under construction in Canada by Dominion Steel Corp. Kaiser Engineers state the oxygen converter process combines the high tonnage-low cost features of the bessemer process with the high quality of the open hearth and is perhaps "one of the most important steelmaking advances in the last 50 years." Steel made by the process has been described by observers in Europe as the best they have ever seen.

One steelmaking authority, while high in praise for the European method, pointed out that there is a difference in the pig iron used in the U. S. and the European pig. Austrian carbonate ores, for example, give a pig iron of about 0.3 pct silicon, while the American average is closer to, or slightly more than 1 pct. This expert feels that double fluxing may be needed in U. S. practice. He also wonders about the effect on furnace linings. It has been reported in European practice the lining is 20 to 24 in. thick and must be replaced or patched after 250 heats.

Linings and fume control play a key part in the Kaiser adaptation of the process. Since attempts at optical measurement of interior temperatures have given 3000° to 3200°F readings thicker linings may be necessary.

In Europe, the huge quantities of dust generated by the process are discharged directly into the atmosphere. Sealed water cooled converters will be used to reduce cost of dust collection methods employed in the States. Air pollution codes will be completely satisfied by entirely enclosed gas control and recovery systems. Another difference in U. S. practice will be pouring and charging from the same side of the furnace.

Current estimates peg capital cost of an oxygen converter shop at 50 pct or less than outlay for a comparably sized open hearth shop—depending on size, location, available facilities, and other variables, Kaiser Engineers say.

One Kaiser estimate on construction costs for a new two-converter plant is said to be about $6.3 million. About $1.9 million of this figure would go for the oxygen generating plant and $990,000 for fume control. The remainder would be for buildings and Bessemer equipment.

George Havas, vice president and general manager of Kaiser Engineers, points out that the oxygen system is relatively simple, consisting essentially of blowing a jet of high purity oxygen upon the surface of molten pig iron bath in a converter vessel. Up to 23 pct scrap can be charged. A wide range of pig iron analyses can be refined by this process and the quality of the resulting steel is as high and as closely controlled as in open hearth practice.

Production is said to be 2 to 3 times higher per hr (about 52 tons per single vessel) than the conventional open hearth furnace. Operating costs, exclusive of metallics and fixed charges, for each ton of steel produced is said to be $3 lower than similar costs for open hearth steel. However, if the process is as good as the various reports say it is, the expert previously referred to sees no reason why finished steel cannot be obtained for an operating cost of $6 or $7 per ton using U. S. pig iron.

C. W. Rueckel, Kaiser Engineers, indicated preliminary talks have taken place with Jones & Laughlin Steel Corp., Weirton Steel Co., and Youngstown Sheet & Tube Co.

Other companies are reported to have patents pending on their own oxygen steelmaking systems. However, another expert points out that almost every conceivable patent for Bessemer steel was applied for and granted during the great Bessemer era of 1865 to 1880.

McLouth Steel Corp., Detroit, may add something to the total picture with an oxygen steelmaking process described as a duplex practice involving the electric furnace in the final stage.

McLouth purchased a 160-ton high purity oxygen plant to serve three 40-ton closed bottom converters which along with a 28-ft hearth blast furnace are scheduled to be installed soon. Metal will be blown to 40 pct of the end point and then charged into 200-ton electric furnaces.

JOURNAL OF METALS, this month, contains three articles dealing with oxygen steelmaking. They are: *The Development of Oxygen Steelmaking*, by Otwin Cuscoleca; *On the Basic Bessemer Process*, by Pierre Coheur and Hans Kosmider; and *European Oxygen Steelmaking is of Far-Reaching Significance*, by H. B. Emerick.

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AT least two things block Japanese titanium production from attaining the large scale output frequently predicted in the country's newspapers—lack of capital and the need for a low cost method for fabrication of titanium sponge, according to *Mineral Trade Notes*, published by U. S. Bureau of Mines. Japan started titanium production on an experimental basis in 1952. Since then the Japanese press has carried numerous articles describing the potentials of titanium and discussing the coming of the new industry.

Progress in terms of actual plant and ouput has been relatively slow. While substantial domestic capital has been attracted to the new industry in the last year and a half, companies who have recently entered the field would welcome additional financing in the form of long term loans from sources outside Japan.

Total estimated output for six plants indicates that some 822 to 970 tons of titanium sponge can be expected in 1954, with 451 to 738 tons in the first quarter of 1955.

Supplies of raw materials, either obtained domestically or abroad, appear to be sufficient for any

foreseeable production goal. No detailed survey of Japan's coastal beach sands deposits have been made, but estimates vary from 100 million to 2 billion tons of ore-bearing sands found mainly on the coasts of Hokkaido and northeast Honshu. Ores used have averaged about 35 pct TiO_2. Costs are high for transporting ore to plants in central Honshu because of the scattered nature of the deposits. Ores are imported chiefly from India, Malaya, and Australia. All Japanese producers use the Kroll process and can produce high quality metal.

Mutual Security Agency, in a 1952 report, termed equipment used by the Osaka Special Iron Works, Ltd., simple, efficient, and adequate. Embassy officials, in another visit to the plant found that the same laborious procedure of dipping out the finished sponge rather than removing the sides of the container is being used in the new 20-ton furnace just completed. The company is limiting its output to sponge. Fabrication is done by the Nagoya Copper Rolling Mill.

A problem faced generally by the industry is determination of a standard titanium slab size for the U. S. market. Other problems besetting Japanese titanium producers besides the lack of capital are the 20 pct import tax imposed by the U. S. on titanium and the difficulty of obtaining certain alloys.

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EDWARD F. Mansure, General Services Administrator, announced that the U. S. Government-owned nickel plant at Nicaro, Cuba, returned $1.1 million to the U. S. Treasury originally loaned to Freeport Sulphur Co. and its subsidiary Nicaro Nickel Co. The money was advanced to permit Nicaro Nickel to gain title to the ore deposits which still feed the plant. At the same time the task of building and operating the huge nickel plant was given to Nicaro Nickel. The original sum was one of the first in a series of investments which have amounted to $50 million. Bulk of the investments is represented by the plant itself.

Nicaro Nickel completed the redemption of 11,000 shares of preferred stock, at $100 per share, which the Government received 12 years ago as token of its stake in the mines, with a final payment of $600,000 in principal and $9600 in dividends figured to May 24, 1954. A total of $1,408,600 was paid by the subsidiary. The excess $308,600 represents dividends cumulated at 4 pct per year since October 1946 on preferred certificates outstanding.

The first payment, $775,000, was made Dec. 31, 1952, less than 12 months after the plant had been restored to operation. Six years' accrued dividends, $275,000, were included in the payment. Dec. 31, 1953, the company paid $24,000 as one year's dividends on the balance. A few months later, by formal notice of intention to redeem the remaining 6000 shares, Nicaro Nickel moved to close the transaction of May 24. Payment was offered and accepted prior to the paid-in-full date.

Treasury reimbursement flowed from royalties which the U. S. paid to Nicaro Nickel for the ore. The company received more than $3.4 million since operations were renewed in January 1952.

Nicaro is currently operated by Nickel Processing Co., whose majority stock is owned by National Lead Co. Production is at an annual rate of 28 million lb of nickel per year. The plant has completed 28 months of uninterrupted operation. A 75 pct capacity expansion has been authorized by the Office of Defense Mobilization and is now being moved from planning boards to the construction stage.

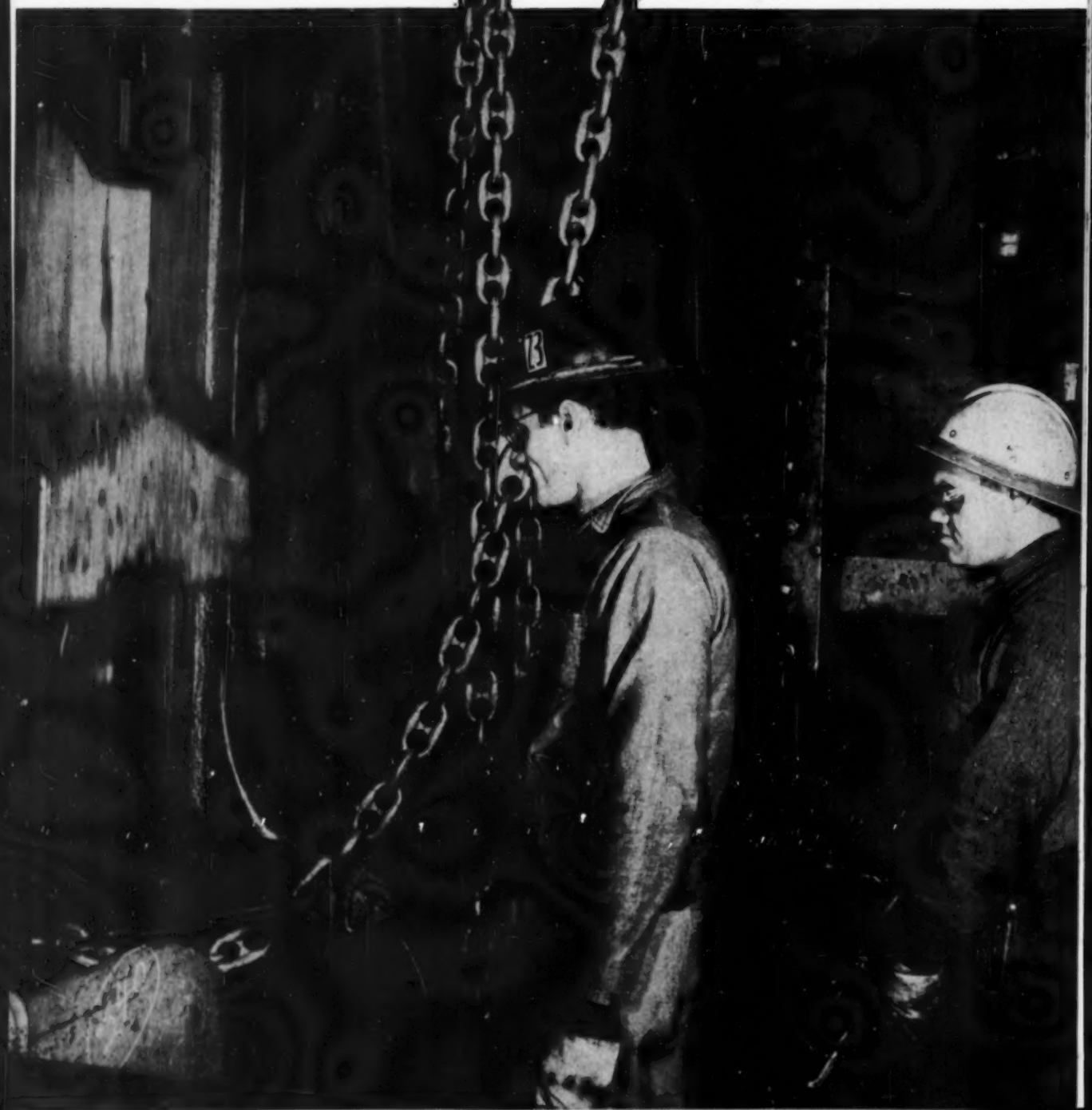
Along with his announcement of the repayment of the original loan, Mr. Mansure commented: "Essentially a nonsubsidized Government operation at present, Nicaro is an asset of growing value to the United States and Cuba. Although this reimbursement is only a small fraction of the Government's total investment in Nicaro, it is a tangible demonstration of the plant's great potential as a going industrial enterprise for Cuba as well as a source of nickel for the United States.

"We will bring the potential nearer reality as we press steadily forward with our plans to enlarge the plant and convert it, by lease or sale, into a permanent commercial venture as soon as it is on a solid financial footing. . . ."

Aluminum Co. of America has placed its 14,000 ton extrusion press in full production at the Lafayette, Ind. Works. It is the first of the giant presses ordered under the U. S. Air Force heavy press program to begin production and is the world's largest extrusion press. The press was originally built in Germany and scheduled for Hitler's war machine. Virtually completed at the end of the war, the press was purchased by Alcoa and redesigned.

Special non-magnetic anchors the

NON-MAGNETIC ANCHOR CHAINS are forged by Baldt Anchor, Chain and Forge Division of The Boston Metals Company, Chester, Pa., from USS CARILLOY austenitic manganese steel. They are air cooled after forging.



USS Carilloy steel chain Navy's newest minesweeper

THE United States Navy has made minesweeping operations more effective with the help of USS CARILLOY steel.

During World War II, a new weapon in marine warfare was developed —the magnetic mine. These mines explode when a ship made of magnetic steel comes near. To sweep such mines safely, without premature or uncontrolled explosion, the Navy is building a fleet of radically new *nonmagnetic* minesweepers.

Finding a suitable steel for the anchor chains on these ships was quite a problem. The chains scrape over hard rocks, sand, and coral along the ocean floor, so they must have good resistance to abrasion. In addition, they must withstand heavy tension, torsion, and impact stresses as wind and tide pound at the ship. To top it off, they must be nonmagnetic and must maintain the non-magnetic feature even after cold working and stressing.

The Navy satisfied *all* these tough requirements with CARILLOY austenitic manganese steel. Intimate knowledge of steels and accurate, skillful control of each step in steel making enabled us to meet the unusual specifications.

Experience on extraordinary jobs like this helps us to do a better job on your steel, whether you need a standard AISI grade or a special analysis. When you need high-quality alloy steel, specify USS CARILLOY. And when you need expert metallurgical advice, get in touch with the nearest USS District Sales Office.

LENGTH of this ¾" anchor chain is checked after proof test loading at Boston Naval Shipyard. Ductility is extremely important in these chains, and USS CARILLOY steel provides an elongation of 80%.

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UNITED STATES STEEL

Drift of Things

PRESIDENT LEO REINARTZ had the misfortune to fall while attending a session at the meeting of the American Iron and Steel Institute in the Waldorf Astoria Hotel in New York on May 26, severely injuring the tendon and muscles in his right leg, just above the kneecap. His leg was operated on by Dr. Preston Wade two days later and he was confined to the New York Hospital for several weeks afterward. He flew back to Middletown on June 17 but does not expect to get about normally before September. (E.H.R.)

ENGINEERS Joint Council has recently published a bulletin on the *Professional Income of Engineers.* (Engineers Joint Council, 29 W. 39th St., N. Y.; price $1.) The information was gathered from questionnaires returned by 295 companies employing 65,169 engineers, supplemented by 12 government agencies employing 3892 engineers, and by returns from 2977 individuals engaged in engineering education. The resultant graphs and data show the upper quartile, median, and lower quartile incomes according to the year of baccalaureate degree.

For all types of engineers, the starting salary paid last year averaged $340 per month; for those working in the field of the AIME, the average was up to $10 more than that figure. Data are consolidated in but two groupings in the mineral industries: 1—"Metal Mining and Primary Fabricated Metal Products Industries", and 2—"Extraction of Crude Petroleum and Natural Gas, and the Manufacture of the Products of Petroleum and Coal." No great difference is evident between these two. The median salary for the first year out of college starts at $4330 for (1) and $4572 for (2); rises to $6423 and $6557 respectively for ten years out of college; and reaches a peak of slightly over $11,000 for the two groups when about 35 years out of college.

The AIME itself has not compiled similar data about its members, but the AIChE last year published results of a questionnaire addressed to its own group, largely chemical engineers of course. For the first ten years out of college the EJC and AIChE data were similar, but after that, the EJC survey showed that salaries began to level off, whereas the chemical engineers' salaries continued to rise. The median salary for engineers that graduated in 1930 was $8780 in the EJC survey, and close to $14,000 in the AIChE survey.

F. J. Van Antwerpen, editor of *Chemical Engineering Progress,* offers some interesting comment on this discrepancy, which applies to AIME:

"Those who made the study for EJC . . . claim only that this survey is valuable in that it reports salaries of *all* engineers and is not 'limited by being restricted, for the most part, to members of professional societies'. Therein is probably the answer for the vast difference in salary experience after the ten-year level.

"Other surveys have revealed this same fact—members of a professional society command higher salaries. Many assumptions can be made as to why, but perhaps it is all summed up by saying that the man who joins a professional society is the man who looks upon engineering as a profession. The man who does not join his professional society looks upon engineering as a job, as a way of earning a living. As so often in life, devotion to a principle brings added things.

"Professional membership must have its influence on the work, the mental attitude, the information quotient, the conversational interests, and the pride of everyone who joins. And over the years, this might just make the difference." (E.H.R.)

WE have just received our first honorary title, which bears with it the privilege of writing F.I.S.H. after our names. Fellow initiates will instantly recognize that as Fellow in the Institute of Sea Horses, headed by Frank La Que, Doctor of Deterioration and Corrosion, and also International Nickel's newest vice-president.

The Sea Horse Institute, however frivolous the name, is an informal organization of people with a pertinent interest in the complex problems of marine corrosion. These people come from all over the country and represent many different industries. Among the AIME members present at the meeting we attended were Norman Cale and Arthur Tracy of American Brass Co., Theodore Monell of Inco, Clarence Merritt of Olin Industries, Sydney Low of Chapman Valve Mfg. Co., John Eckel of General Electric Co., and Olof Dormsjo of A. Johnson & Co. Highest ranking Naval officer attending was Rear Admiral Logan McKee, Commander of Norfolk Naval Shipyard. Inco's Bob Johnson served as master of ceremonies and host.

Scene of the activities was Wrightsville, N. C., location of International Nickel Co.'s corrosion testing stations at Harbor Island and Kure Beach. The purpose of our visit really was a serious one, attending the Ninth Annual Engineers Group Inspection and Conference. With the nation's annual loss due to corrosion estimated at $6 billion, there is much that must be done in addition to all that is being done to bring this figure down. Basic interests at Harbor Island are sea water corrosion of all types of materials, and atmospheric sea spray corrosion tests are conducted at Kure Beach. The Inco Marine Laboratory is definitely not limited to the products of that company, but conducts tests for other producers of metallic and nonmetallic materials, and does much work for the government. The test results are eventually made available to all industry.

This was the first technical conference we had ever attended that was conducted by an industrial concern, and it was well organized. Sessions always started on time, and that's more than we can say for some of our own meetings! It was also very thoughtful of Inco to locate their test station in the midst of a lovely resort area with excellent beaches. We had the opportunity to do a little corrosion experimenting, and our results showed that a little suntan is a good coating for protection during sea water immersion and atmospheric sea spray tests.

A. S. Cohan

(Refractory Oxides) → (Complex low melting oxides)

which . . . translated with liberties . . . **means: 1 ton of dirty steel** plus **10¢ worth of High Carbon Ferro Titanium** yields **1 ton of clean steel**

The small amount of alloy used (generally 1 lb. per ton) is the key to effective scavenging action for killed and semi-killed carbon and low alloy steels. Titanium dioxide (TiO_2) formed from the addition is light in weight, an excellent flux for refractory oxides which form complex oxides which are "sticky" in the fluid state at the temperature of molten steel. As these low melting point oxides rise, they pick up remaining non-metallic inclusions and carry these inclusions to the slag in the ladle.

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Ideal for mildly corrosive service

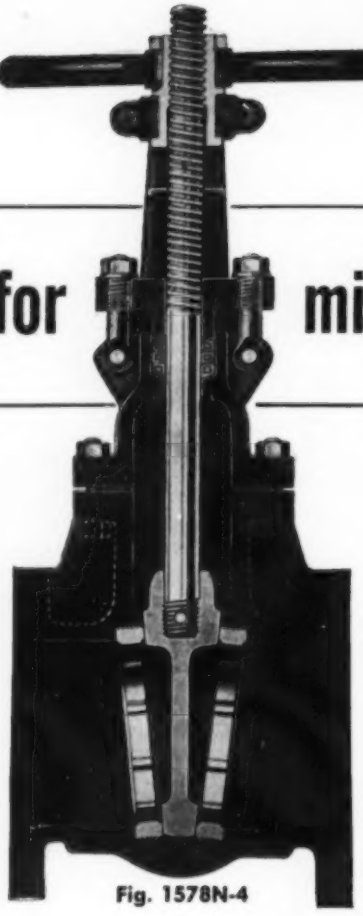


Fig. 1578N-4

3% Nickel Iron Gate Valve, Class 125 lb. . . . for process industries . . . developed and produced by The Lunkenheimer Co., Cincinnati 14, Ohio. Outside screw and yoke, bolted bonnet, flanged . . . face to face dimensions are American Standard for 125 lb. Cast Iron Wedge Gate Valves (ASA B16.10-1939).

LUNKENHEIMER develops new 125 lb. nickel-iron valve for process industries...

Mildly corrosive valve service poses a particularly tough problem in the process industries. To get *maximum* corrosion resistance, you have to pay for more than you need—high-alloy valves are expensive. And yet, ordinary cast iron often won't do the job.

An economical answer to many of these problems has been developed by Lunkenheimer with this new line of Nickel-Iron Gate Valves. Bodies and bonnets are 3% Nickel-Iron, for better corrosion resistance than ordinary cast iron, and trim is offered in choice of 18-8 (Type 316) Stainless Steel or Monel®, for maximum resistance at the point of most severe service. This combination is cutting valve costs in the petroleum field, pulp and paper, wood treating, and scores of other process industries.

This development is but one more instance that shows how nickel alloys can be used to provide significant advantages, whatever the equipment.

Many standard nickel alloyed compositions are available, making it easy to select one with extra qualities for almost any specific use. Whatever your industry, if you have a metal problem, send us details for our suggestions. Write today.

EMBLEM OF INCO SERVICE

THE INTERNATIONAL NICKEL COMPANY, INC. 67 WALL STREET NEW YORK 5, N. Y.

Sherritt Gordon Mines received $1.42 million from the U. S. as an advance on payment for recoverable nickel content of concentrates stockpiled at the Fort Saskatchewan refinery. Cost of the Sherritt Gordon project to bring Lynn Lake, Manitoba nickel properties into production exceeded original estimates by 43 pct.

Substitute for African palm oil used in making hot dipped tin plate and in cold rolling thin sheet and strip has been uncovered through research by the American Iron & Steel Institute. The substance is derived from beef tallow.

A small piece of glass coated with lead sulphide may prove invaluable in developing devices for detecting warm objects at great distances. Produced by Eastman Kodak Co., the Ektron detector is said to have 10,000 times the sensitivity to certain infrared rays as previous laboratory instruments. In addition to heat detection devices, the sensitive cells may be applied to switching devices.

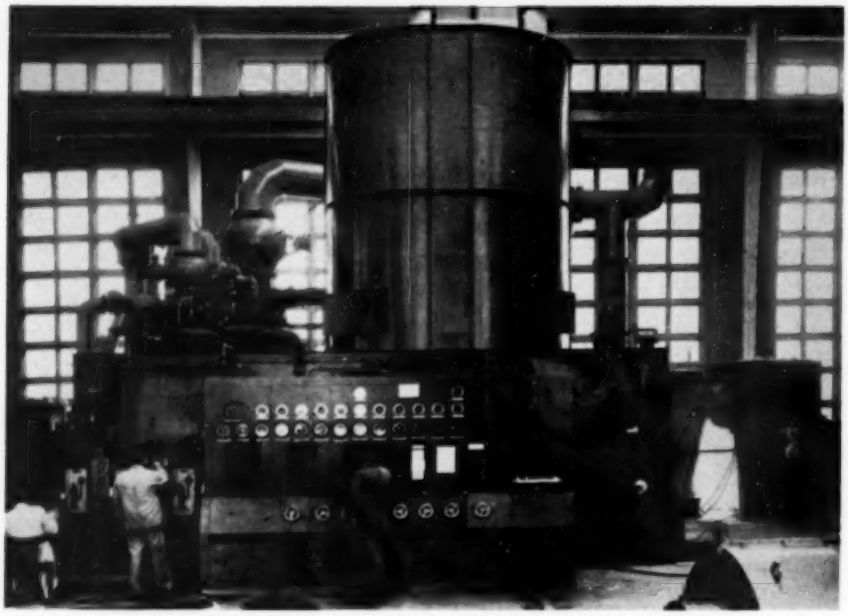
It is reported that a preliminary "letter agreement" had been reached between the U. S. and Bolivia for 12,000 tons of tin in concentrates in anticipation of continued operation of the Texas City tin smelter. At the time the letter was written, Congress had made no final decision on the fate of the smelter.

The new 20 ton automatic double melting furnace designed by Titanium Metals Corp. of America completely remelts 4000 lb titanium ingots with improved metal homogeneity. The vacuum melting process, instead of the usual method of flooding the furnace with slightly positive pressures of argon or argon-helium mixtures, allows removal of hydrogen from titanium metal down to as low as 0.005 pct or 50 parts per million.

Metals are flowing steadily from free world nations to iron curtain countries, through various European free ports, according to a report on transit trade published by the Dept. of Commerce. In 1953, Soviet Russia received 13,386 metric tons of lead through Antwerp.

Contracts with Anaconda Sales Co., for purchase of 64,000 tons and Kennecott Sales Corp. for 36,000 tons of Chilean copper conclude total purchase agreement of 100,000 tons signed with the Government of Chile. The price is 30¢ per lb including delivery to storage points, less allowances for cathode copper.

Officials of Dominion Magnesium Ltd., sole Canadian producer of titanium metal powder, think that prospects for producing an acceptable grade of metal for forgings and sheet have improved. Pilot plant operations indicate that metal can be produced at an attractive price.



Blaw-Knox Co.'s 190-ton per day Linde Fraenke oxygen plant undergoes initial test operations.

European Oxygen Steelmaking Is of Far-Reaching Significance

by H. B. Emerick

DURING a visit to several of the leading steel plants and equipment manufacturers in Germany, Austria, Switzerland, Belgium, and France in July 1953, the most outstanding technological development observed was the widespread and steadily expanding application of tonnage oxygen to iron and steel production processes.

Mathias Frankl's discovery of replacing the tubular recuperator with thermodynamically efficient reversible regenerators in the liquefaction and separation of gas mixtures greatly simplified the process of producing metallurgical oxygen. As a result, virtually every integrated steelworks in Western Europe has installed facilities for producing high purity oxygen in large quantities at low power costs.

The principal European applications of oxygen as an aid in metallurgical processes are as follows: A—Enrichment of compressed air used in blast furnace operation; B—enrichment of air blast in bottom blown basic bessemer practice; C—prerefining of basic iron in the hot metal ladle; D—nitrogen free mixtures of high purity oxygen and steam in basic bessemer practice; and E—top blowing in solid-bottom basic lined converters.

H. B. EMERICK is Assistant Director—Technical Services, Jones & Laughlin Steel Corp., Pittsburgh. This paper was presented at the AIME National Open Hearth Conference, Chicago, Apr. 5 to 7, 1954.

Oxygen in Blast Furnace

At least three experimental low-shaft blast furnaces in Belgium and Germany are designed for enrichment of the blast to a maximum level of 50 pct O. Enriched blast appears to be especially advantageous in cases where ferroalloys are smelted from low grade ores using relatively inferior carbonaceous fuels. No important applications of oxygen were observed in connection with the smelting of steelmaking iron in the traditional high-shaft blast furnace.

Oxygen in Thomas Converter

The importance of this process in the steel industry of Western Europe is evidenced by the fact that Thomas steel makes up 40 pct of the German production, 60 pct of the French production, 85 pct of the Belgian production and practically all of Luxemburg's. It is not surprising therefore to note intense interest in finding ways to improve the process. One source of difficulty has been the relatively high nitrogen content which is characteristic of ordinary Thomas steel. Consequently, several plants are now using air blast enriched to 30 to 35 pct O as standard

practice in order to gain important advantages in shorter blowing time, lower nitrogen content and increased scrap melting capacity. The oxygen addition is usually made during the decarburization period.

Preblowing in Hot Metal Ladle

It has been a matter of experience in Europe when bottom blowing high silicon iron with oxygen enriched air that metal is frequently ejected from the converter during the early, low temperature stage of the blow. To overcome this difficulty a method has been developed to desiliconize hot metal in the iron ladle with low pressure oxygen, and thus produce a superheated iron which can be successfully bottom blown with oxygen-enriched air at full blast throughout the entire refining period. The net result is an attractive further reduction in blowing time and still lower nitrogen content in the finished steel.

In one Ruhr district plant approximately 80 pct of all iron produced is preblown with oxygen before going to the mixers. There are six blowing stations under one roof, each with a water cooled cast iron hood rigged so it can be lowered flush against the rim of the iron ladle. Projecting laterally through a hole in the side of the hood is a short water cooled, copper tipped oxygen lance tilted downward at a 30° angle so that the tip is about six in. below the surface of the metal. Oxygen of 73 pct purity is blown through the 0.4 in. orifice for about 30 min at a pressure of 2.5 atm. The effect of this treatment is to reduce the silicon content about 0.25 pct and the manganese about 0.60 pct, with a 100° to 150°F rise in iron temperature. Carbon and phosphorus are not affected. The slag formed is foamy but relatively inert since no vigorous boiling action takes place. A major problem associated with oxygen preblowing is the collecting and cleaning of the dense oxide fumes which are generated in the process.

Steam and Oxygen Mixtures

To achieve the absolute minimum nitrogen content in Thomas converter practice, it is necessary to eliminate nitrogen entirely from the refining gas. But pure oxygen cannot be used in a bottom-blown converter because the refractories in the tuyere area would not stand it. Therefore, another inert gas must be substituted for the nitrogen component of ordinary air. Steam (water vapor) is the most economical gas for dilution. This combination is used successfully in Belgium and Germany where converters are being blown continuously with a mixture of 60 pct O (95 pct pure) and 40 pct superheated steam. This mixture is approximately thermally equivalent to air, but some scrap melting capacity is sacrificed. By this method it is possible to produce steel having a maximum nitrogen content of 0.0025 pct. Oxygen retained in the steel is not abnormally high, and hydrogen absorption apparently has given no trouble.

Oxygen Steelmaking Process

This process, which is evolutionary in character rather than revolutionary, combines certain advantages offered by both the basic open hearth and the basic converter processes, and at the same time eliminates many of their respective deficiencies. In this process molten blast furnace iron is refined to steel in a basic lined, solid (tuyereless) bottom vessel by directing a jet of nearly pure oxygen vertically, at high velocity, onto the surface of the hot metal bath. The bath absorbs the gas almost completely, as the actual oxygen consumption is very close to the theoretical requirement. Oxygen of 98 pct minimum purity is discharged through a retractable water cooled, copper-tipped lance. The resulting product—low in phosphorus, sulphur, and nitrogen contents—is comparable in quality with basic open hearth steel, yet the production rate is on a level with the conventional bessemer converter processes. The capital investment and unit conversion costs are much lower than for an open hearth shop of equal capacity. Actually, the cost of installation is no higher than for converter plants of traditional design since the additional cost for a tonnage oxygen production plant is offset by the elimination of auxiliary equipment such as turboblowers, bottom house and bottom changing facilities.

A major advantage of the oxygen converter process is its remarkable flexibility in handling a wide range of raw materials. The process can be operated on any kind of hot metal that can be used in the open hearth furnace and in addition it will accommodate iron of a type which is normally unsuitable either for the open hearth or the conventional converter processes. Hot metal mixers are employed to insure satisfactory uniformity of iron composition for best temperature control.

The recently completed oxygen steelmaking plants at Linz and Donawitz (Austria) operate on blast furnace metal which covers a range from 0.2 to 1.5 pct Si, from 1.5 to 3.0 pct Mn, and up to 0.25 pct P. In Germany the process is being successfully adapted for use with Thomas pig iron containing up to 2.0 pct P. It has been discovered that dephosphorization can be made to take place much earlier by the addition of iron ore, which apparently acts not only in a physical sense by cooling the bath, but also chemically by promoting an earlier slag formation. Moreover, the use of ore provides a metallurgically effi-

The oxygen steelmaking process combines the high tonnage, low cost features of the conventional Bessemer shown above with the high quality of the open hearth.

cient method of direct reduction of crude ore to liquid steel.

Depending on the level of heat producing elements in the iron, the normal range of scrap additions in the two Austrian production shops is 12 to 25 pct of the gross metallic charge. This is higher than has been possible in bottom-blown converter practice.

Refractories

The converter itself may be either eccentric or concentric in design. A permanent lining next to the shell plates consists of a single course of burned magnesite brick. On top of this is a rammed layer of tar-bonded dolomitic magnesite which is renewed each time the vessel is relined. The inner or working lining is a very thick, unfired brick pressed from a mixture of sintered magnesite and coal tar pitch. The lining thickness is approximately 20 to 24 in.

The solid bottom of the vessel, which may be either integral or detachable, is somewhat hemispherical in contour and is built up much like a basic open hearth bottom. There is a single course of clay brick on the shell, then two horizontal courses of magnesite brick, a soldier course of magnesite brick and, finally, a rammed layer of burned magnesite to form the working surface. The bottom is about 26 in. thick overall. Normally it remains in good condition indefinitely, while the body lining fails by even wear of the side walls in the reaction zone. The lining must be patched or replaced after blowing approximately 250 heats or roughly after one week of operations.

Method of Operation

The blowing equipment consists of a tubular water cooled, retractable jet device kept in a vertical position above the center line of the vessel. At the top of the lance, armored rubber hoses are connected to a pressure-regulated oxygen source and to a supply of recirculated cooling water. After charging the vessel with scrap and hot metal, the oxygen jet is lowered to a position several inches above the surface of the bath. This distance is fixed by experiment and certain known variables. Oxygen issues from the jet nozzle under pressures which may be regulated over a fairly wide range but are usually held between 100 and 150 psi. Impinging on the surface of the liquid bath the oxygen instantaneously starts reactions leading to the formation of FeO, part of which diffuses rapidly through the bath. Carbon monoxide is immediately evolved which gives rise to a vigorous boiling action and accelerates the refining reactions.

Slag forming fluxes, chiefly burned lime, are adding during the blow through inclined chutes built into the sides of a brick lined smoke hood that covers the vessel. The hood is connected to a stack which discharges waste gases through the roof to the atmosphere. No flame is visible outside the mill buildings. A dense cloud of reddish-brown fumes is emitted from the stack. The dust is mainly iron oxide of extremely fine particle size, which complicates the gas cleaning problem.

All heats are blown down to 0.05 or 0.06 pct C and recarburized in the ladle if necessary. The endpoint normally occurs after 18 to 20 min and is signalled by the collapse of the carbon flame. Gross yield, from metal charged to liquid steel, is approximately 90 pct.

Since the oxygen steelmaking process employs a refining agent whose nitrogen content is practically nil, a steel is produced which is distinguished by its exceptionally low nitrogen content—generally 0.004 pct or lower and well within the expected range for open hearth steel. Eighty percent of the heats finish below 0.030 pct S and 0.025 pct P. The material exhibits good cold forming properties and good weldability. In a wide variety of applications including wire products, tin plate, hot and cold rolled sheets, and plates and bar products the material has held its own against basic open hearth steel of similar carbon content.

Considering that pilot scale experiments started less than six years ago, the oxygen steelmaking process has experienced phenomenally rapid development to its present established position as a mature commercial process. In 1953 the process accounted for 300,000 metric tons of steel production in Austria alone, or 22 pct of that country's output. In 1954 this will be expanded to 400,000 metric tons. By 1955 it is estimated there will be at least 1,000,000 metric tons of installed capacity here and abroad.

Any steel company planning a carbon steel ingot expansion program, or seeking a way to improve upon existing converter steelmaking methods, cannot afford to overlook the oxygen process.

Summary

The most significant and far-reaching postwar development in European steelmaking practice stems from the discovery and application of new means for producing tonnage oxygen at low cost by liquefaction of air. Much has been accomplished in Europe toward developing new and efficient methods for utilizing pure oxygen in a variety of metallurgical processes. The most important of these in the field of ferrous metallurgy are: A—Enriched air blast for low-shaft reduction furnaces; B—enriched air blast for bottom-blown basic bessemer converters; C—prerefining of iron in the hot metal ladle—applicable to either basic open hearth of basic bessemer operations; D—oxygen and steam mixtures for bottom blowing in basic bessemer converters; and E—oxygen top blowing in solid bottom converters. The last of these developments seems destined to command worldwide attention as it gives every promise of placing the pneumatic process of steelmaking in an entirely new perspective.

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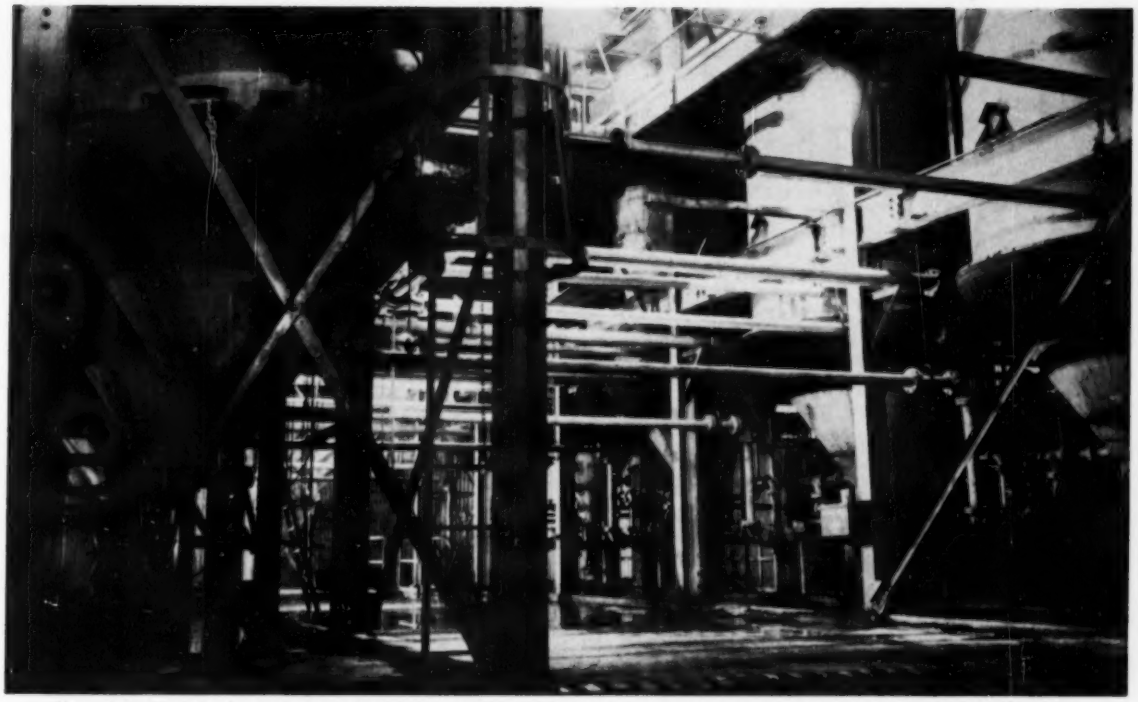

One of the steps in the extraction of titanium metal using a chloride process is the separation of titanium metal from magnesium and magnesium chloride in acid leaching tanks, shown in this picture of the TMCA plant in Henderson, Nev.

Titanium Extraction by Chloride Process Presents a Variety of Problems

by W. R. Opie

PROCESS metallurgists face a number of problems in attempting to extract titanium metal from its ores with a chloride process. The steps in this process can be considered as: A—Preparation of a chlorinator feed; B—chlorination to produce $TiCl_4$; C—purification of $TiCl_4$; and D—reduction of $TiCl_4$ to titanium metal.

There are three groups of possible ores or raw materials available in large quantities as starting material for a chloride process. The purest and most expensive are commercial grades of titanium dioxide. The next highest grade material is rutile ore, and the most abundant and cheapest materials are the titaniferous ores, ilmenite and titaniferous magnetite. The approximate cost per pound of contained titanium in each source is: "TITANOX" TG at 42¢, rutile ores at 8¢ to 16¢, and ilmenite at approx 3¢.

As can be seen ilmenites are the cheapest. If the iron can be recovered from them in a salable form, the actual cost of the contained titanium will be reduced by the iron credit making the cost differential between rutile and ilmenite even greater.

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Rutile Chlorination

If a high purity oxide or rutile ore is used as feed many problems are eliminated. Economics, however, rule out the pure oxides, and at present the titanium metals industry is almost entirely concerned with actual metal production from rutile ores. Although rutile usually occurs disseminated throughout sand deposits it can be readily concentrated to a product shown in Table I.

Although rutile is a high grade ore the impurities that it does contain are a source of trouble during the chlorination step. The interfering elements as far as chlorination of any titanium containing ore is concerned are classified in Table II. It can be generally stated that anything that goes into the chlorinator in addition to the titanium values can cause trouble to a degree.

The inerts, which are difficult to chlorinate, interfere in the chlorination process to a minor extent only. They absorb some heat and remain as a residue which must be removed in some manner from the chlorination system or from the resulting tetra-

Table I. Typical Composition of Available Rutile

Material	Pct	Material	Pct
TiO_2	97.0 to 98.5	Al_2O_3	0.05 to 0.1
FeO	0.1 to 0.5	ZrO	0 to 1
Fe_2O_3	0.1 to 0.5	Nb	0.2 to 0.4
MgO	0	V	0.2 to 0.6
CaO	0	Cr	0.2 to 0.4
SiO_2	0.1 to 1.0	Sn	0 to 0.2

chloride where they exist as fine particles in suspension. Silica and alumina can come from the reducing agent, coke or coal, as well as from the ore.

The volatile chlorine demanders cut down chlorine utilization efficiency and contaminate the resulting tetrachloride, making subsequent purification necessary.

Nonvolatile chlorine demanders also decrease chlorine efficiency and tend to slow up the chlorination reaction by coating the ore particles. If present in large quantities they cause bed sintering making chlorination inefficient. The chlorides in this group volatilize to a slight extent so that a small percentage can end up in the resulting tetrachloride.

From the purification standpoint the nonvolatile chlorides, fine particles of inerts, volatile chlorides which are insoluble in $TiCl_4$, and unreacted reducing agents interfere by remaining in suspension in the liquid tetrachloride. These present a filtering or settling problem which cannot be ignored.

The volatile chlorides which are soluble in the titanium tetrachloride, including oxychlorides of titanium, can in most cases be easily separated by proper fractional distillation. Some of them, however, have boiling points very close to that of titanium and must first be changed in chemical composition so that they will not carry over with the titanium tetrachloride.

Rutile is used extensively as a chlorination feed and problems of handling its interfering elements have been faced and for the most part solved although many refinements in techniques can be developed which will make manufacture of a high purity product more efficient.

Other Chlorinator Feed Material

Other titanium ores which will come into the titanium metal picture are the ilmenites and titaniferous magnetites. Ilmenites will probably be used for one or all of the following reasons:

A—Rutile sources will not be sufficient to keep up with rapidly expanding production.

B—Metal producers will develop a means of handling ilmenite so that it can compete with rutile.

C—Pigment producers will develop processes for making a cheap high TiO_2 content feed material from ilmenites.

The incentive is real for utilizing the titaniferous ores and much effort is being expended toward their beneficiation. They can be classified roughly into three main categories: A—High titania ilmenites containing more than the theoretical ilmenite TiO_2 content (52.7 pct); B—low titania ilmenites containing less than the theoretical TiO_2 content; and C—titaniferous magnetites. Table III lists some typical ore compositions in these three categories.

If an ilmenite is considered as a feed to a chlorinator the problem of handling the large amount of iron, magnesia, calcium oxide, silica, and alumina must be faced. The iron is objectional for three main reasons: A—It is a chlorine demander; B—it forms $FeCl_2$ which tends to clog off-take lines; and C—it is a sludge former in the condensed $TiCl_4$.

Table II. Interfering Elements in the Chlorination of Titanium Ore

Chlorine Demanders		Inerts
Volatile Chlorides	Nonvolatile Chlorides	
$FeCl_3$	$FeCl_2$(partly)	SiO_2(partly)
$ZrCl_4$	$MgCl_2$	Al_2O_3(partly)
$NbCl_4$	$CaCl_2$	
VCl_4	NaCl	
$CrCl_3$	KCl	
$SiCl_4$		
$AlCl_3$		
$MnCl_2$		

In order to develop an efficient process to chlorinate ilmenite directly, then, the chlorine used by the iron must be recovered by breaking down the $FeCl_2$ in a separate operation and the large amounts of this high volume per unit weight material must be handled in some ingenious manner during the chlorination phase so that the $TiCl_4$ off-take lines are kept open. These are difficult problems to solve and other approaches have therefore been considered by which the iron can be removed from the ilmenite before it reaches the chlorinator. Some typical lines of approach are: A—Mineral separation with and without previous treatment; B—chemical removal; and C—smelting.

It becomes quite obvious through petrographic examination that the majority of the iron is tied up in the same mineralogical phase as the titania.

Table III. Composition of Titaniferous Concentrates, Pct

Material	High Titania Ilmenites		Low Titania Ilmenites		MacIntyre Magnetite
	Quilon*	Florida	MacIntyre**	Bale St. Paul***	
TiO_2	57.5 to 61.0	58 to 63	43 to 50	38.5 to 41.5	8 to 11
FeO	9 to 19	4 to 8	32 to 39	28 to 31	30 to 34
Fe_2O_3	20 to 26	22 to 28	5 to 14	18 to 21	46 to 50
MgO	0.5 to 1.2	0.4	2 to 4	3.5 to 4.5	1.5 to 3.0
CaO	0.1 to 0.2	0.5	0.5 to 1.0	0.3 to 1.0	0.6 to 1.5
SiO_2	0.5 to 2.0	0.5 to 2.0	1 to 5	2 to 9	1.5 to 4.5
Al_2O_3	1 to 2	1 to 3	1 to 4	2 to 4	3 to 6
ZrO	0.5 to 2.0	0.2 to 0.4	0.01 to 0.05	0.0 to 0.05	—
Nb	0.01 to 0.15	0.01 to 0.06	<0.01	<0.01	—
V	0.05 to 0.4	0.04 to 1.0	0.01 to 0.05	0.1 to 0.2	0.6 to 1.0
Cr	0.05 to 0.2	0.08 to 0.2	0.01 to 0.04	0.1 to 0.3	2.0 to 0.4
Mn	0.05 to 0.4	0.3 to 1.0	0.2 to 0.5	0.1 to 0.3	0.4

* Brazil. ** Tahawus, N. Y. *** Labrador.

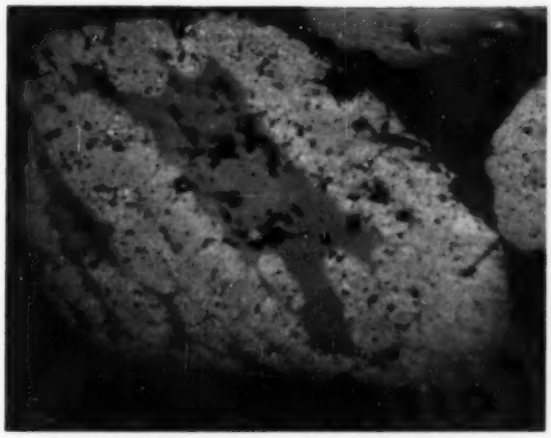


Fig. 1—Partially altered grain of Quilon (Brazil) ore is shown magnified 1000X. The fresh ilmenite ore is dark gray and irregularly rimmed by light gray granular alteration products.

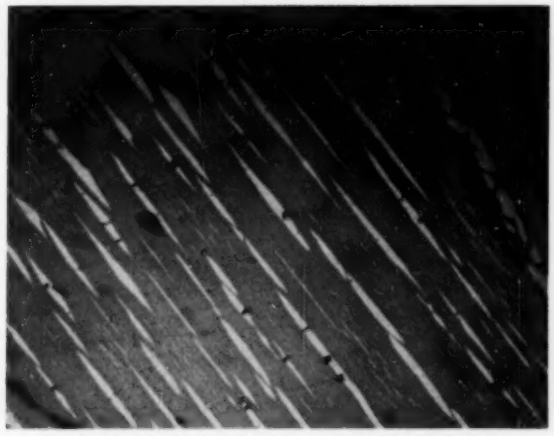


Fig. 2—Baie St. Paul (Labrador) ore magnified 250X shows exsolution lenses of light hematite in coarse dark ilmenite. This close iron-titania association is the basic mineral dressing problem.

Grouping of titaniferous material as shown in Table III is convenient for discussing TiO_2-Fe associations.

The high titania ilmenites such as Quilon (Brazil) and Florida types have undergone some form of weathering, either hydrothermal or from reaction of ground and surface waters. Fig. 1 shows a typical grain of Quilon beach sand that has undergone some weathering. Unaltered ilmenite is in the center with the alteration products surrounding it. As disclosed by Lynd, et al[1] the effect of the weathering has been to break down the ilmenite structure leaving very fine closely associated Fe_2O_3 and TiO_2.

The low titania ilmenites such as the MacIntyre (New York) and Baie St. Paul (Labrador) types also have a close iron-titania association. The iron oxides in excess of the amount in solution in the ilmenite structure (about 48 pct FeO) occurs in lathes in the ilmenite as shown in Fig. 2. The iron and titania are also closely locked in the titaniferous magnetites, the titania occurring as ilmenite lathes in the magnetite.

The best, then, that can be hoped for from mineral dressing techniques in the way of upgrading in TiO_2 content any of the materials in Table III, which are actually concentrates from some mineral separation process, is to remove a little more of the gangue which is already close to the economic lower limit.

Various reported treatments coupled with a mineral dressing step have shown some promise in breaking down the ilmenite structure and removing a portion of the iron. A solid state reduction of the iron followed by fine grinding and magnetic separation is one approach. The problem is to get the reduced iron to agglomerate into particles large enough so that it can be separated. The reoxidation of the fine iron particles during a crushing step must also be coped with. With such a treatment a product low enough in iron to be handled in a chlorinator might possibly be produced.

For many years the titania in ilmenite has been separated from the iron by digesting the ore in strong sulphuric acid to solubilize both the titanium and iron and subsequently precipitating the titanium selectively as the hydrate. The iron is a waste product and takes with it valuable sulphuric acid. Any chemical process for separating the iron from titania, to be attractive, should use a cheap reagent or a reagent that can be recovered and reused. Any way of yielding a salable iron byproduct will help the economics.

Smelting

A most attractive way of producing a high titanium concentrate is to smelt ilmenite or titaniferous magnetites to yield a titania slag and a salable iron. The Quebec Iron & Titanium smelter at Sorel, Que. has been furnacing Allard Lake ore which is similar to the Baie St. Paul ilmenite in Table III. The resulting slags have approximately the composition shown in Table IV.

Although this material can be chlorinated under certain conditions it is not an ideal chlorinator feed and work is progressing to yield a furnace product higher in TiO_2 with less of the troublesome oxides necessary to make this slag fluid.

There are a number of approaches which can be taken to make a slag equivalent or better than natural rutile. Each avenue has the same goal, to produce a product which can be economically furnaced and which will be low enough in chlorine demanders to be economically chlorinated.

A slag composition as shown in Table IV is just about fluid enough to be handled in a conventional electric furnace. No fluxes are necessary to make this composition from Allard Lake ores. However, if the FeO content is lowered a fluxing oxide must be added such as lime or soda to keep the slag fluid enough to pour. This approach yields a slag low enough in iron for efficient chlorination but too high in lime, soda, and magnesia which tend to form chlorides which coat the slag particles and prevent effective chlorination of the titanium values.

If ores such as Quilon and Florida which are low in gangue are used, and no fluxes are added which will interfere with chlorination, the FeO content must be kept high to maintain fluidity. Such a slag would be much too high in iron to be a good feed.

Table IV. Typical Composition of Sorel Titanium Smelter Slag

Material	Pct
TiO_2	70
FeO	10
MgO	5 to 7
Al_2O_3	8 to 10
SiO_2	5 to 7

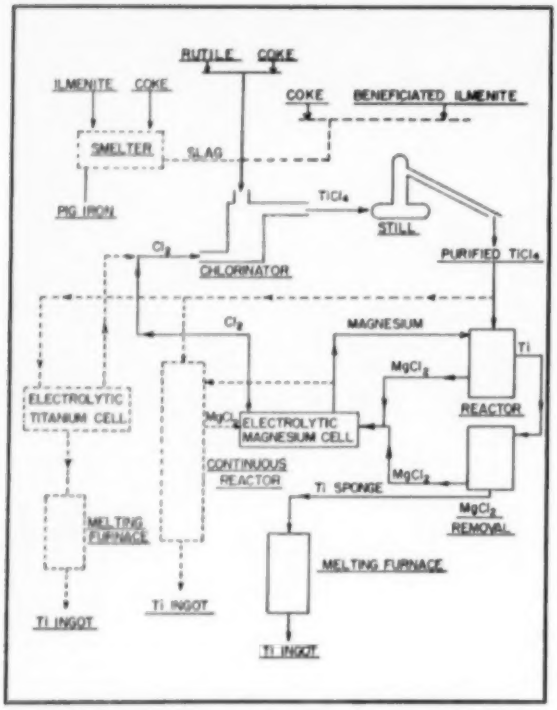


Fig. 3—If a chloride process becomes the accepted one for large scale titanium production it will probably look like one outlined above. From left to right are shown the electrolytic chloride, continuous magnesium reduction, and batch magnesium reduction processes.

The production of a slag that can be easily chlorinated, then, is no simple procedure and much work is being done and will be done along this line. The reward is to get a titanium source for nothing, letting the iron pay for the smelting operation.

Production of $TiCl_4$

The main problems in the reduction step are: A—To prevent air contamination during the reaction and during sponge removal; B—to separate efficiently the products of the reduction before or during sponge melting; and C—to prevent contamination from the reaction container.

Air or oxygen, either of which destroy ductility if present as dissolved oxygen or nitrogen in titanium metal, can enter a reactor or electrolytic cell through leaks in the container walls as dissolved gases in the $TiCl_4$ or as dissolved or suspended oxides and nitrides in a reducing metal if one is used. Care must be taken to prevent oxygen or nitrogen contamination by employing vacuum or inert atmospheres in the reactor. Merely maintaining a good vacuum is obviously not enough. The reactor must be *leak free* or the vacuum pump will serve only as a means for pulling a supply of fresh air through the reaction zone. If inert atmospheres are used a *leak free* condition is also essential. Even though large gas containers are kept under positive pressure back diffusion of a contaminating gas can take place through small openings especially if pressure fluctuations occur, as they often do, caused by turbulent flow of gas within the container as a result of a temperature differential.

Indications are that $TiCl_4$ dissolves air if it is allowed to contact it. Any moisture in the air will quite obviously react to form HCl, TiO_2, and probably oxychlorides. These products are removed during a refining step. There is some solubility of dry air, however, which can be troublesome. Some equilibrium solubility data and a reliable method of analyzing for air would be valuable.

If magnesium is used to reduce $TiCl_4$ the problem of preventing introduction of oxygen into the reactor as magnesium oxide must be coped with. The density differential between magnesium and its oxide is so slight that separation during melting and refining is difficult.

Problems encountered in the electrolytic reduction of $TiCl_4$ are also complex. The four valent chloride is not readily soluble in alkali or alkaline earth chlorides so the first step must be to get the titanium values in solution by reducing or complexing the $TiCl_4$. Once the titanium values are in the electrolyte as divalent or trivalent titanium the electrolysis must be carried out so that energy consuming oxidation-reduction reactions from tri to di at the cathode and di to tri at the anode are kept at a minimum. After the deposit is formed it must be removed through a suitable mechanism which prevents entrance of air into the cell. High temperature electrolysis to form a molten metal at the cathode is extremely difficult because of the problem of obtaining a suitable nonreactive container to hold the electrolyte and the molten metal. Such a cell would have to be operated at about 1800°C.

Many of the problems discussed have been solved so that commercial production of titanium is a reality. Work is progressing steadily to improve present processes and develop new ones. It is probable that if a chloride process becomes the accepted one for large scale production it will look like one of those outlined in Fig. 3.

The present batch magnesium reduction process is outlined on the right of Fig. 3, using a rutile feed material. This process has more experience behind it and will probably give any process competition for many years to come.

The steps of a continuous magnesium reduction process are outlined in the central part of Fig. 3. Obviously a continuous reactor which would turn out a metal compact or an ingot would have many advantages, mainly much greater output per unit.

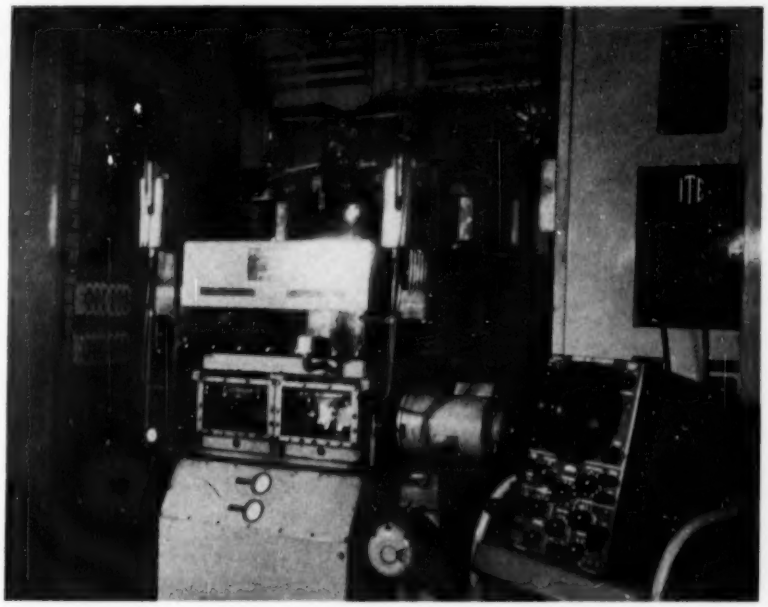
An electrolytic chloride process as shown in the left part of the drawing would eliminate one major step—magnesium production. At the present all three types, and probably others, are being considered.

Conclusion

The purpose of this presentation will have been served if it makes clear the many complex problems facing the process metallurgist in attempting to make a ductile titanium sponge from its ores. Rather than take the viewpoint of the physical metallurgist or metal fabricator that "the problems are only beginning when you get the sponge," the process metallurgist prefers to think that the battle is more than half won when a high quality sponge is delivered to the melting furnace.

Reference

[1] L. E. Lynd, H. Sigurdson, C. North, and W. W. Anderson: Chemical and Mineralogical Characteristics of Titaniferous Concentrates from North Carolina, Quilon, Brazil, and Florida Beach Sands. (Unpublished manuscript.)



Interior view of the mechanical rectifier cubicle shows main contacts and synchronous motor driven camshaft in foreground. Tap changer, motor driven ac disconnects, and short circuiting switches are in rear. Monitoring oscilloscope is at right.

Direct Current Requirements At Carteret Copper Refinery Supplied by Three Units

by Edwin M. Meyer

FROM the viewpoint of electric power supply the electrolytic copper refining operations at Carteret may be divided into three general classifications each with its own peculiar electric load characteristics. These are: A—Regular or commercial deposition, B—copper liberation, and C—copper powder deposition.

A commercial refining section consists of a battery of depositing tanks (usually 38 tanks) each containing 37 pairs of spaced electrodes plus an extra cathode. That is, 37 cast anodes made up of the furnace product to be refined and 38 cathode starting sheets on which the refined copper is deposited. Electrode pairs are electrically in parallel within a single tank. Tanks are electrically in series within a single section, and several sections (usually 8 to 12) are electrically connected in series to form a *tank house circuit* which is supplied by its own individual generator.

It is not the practice at Carteret to connect tank house circuits in parallel on a common supply bus because of a type of operation requiring one or more daily changes in the number of sections in circuit with resultant variations in total circuit voltage. However, the common bus system, fed by one or more generators in parallel is used in other refineries where operating conditions will permit.

Cathode starting sheets on which the deposition of refined copper is initiated are made in modified commercial sections in which the starting cathodes are oiled plates of refined rolled copper. Copper deposition is allowed to build up to approximately 0.025 in., then stripped off at the oiled surface to form a starting sheet. This sheet is fitted with hanging loops of the same material and is transferred to a commercial section to be built up into a regular cathode approximately $\frac{5}{8}$ in. thick. These modified sections are called *stripper* sections. These differ (as far as the electrical features are concerned) only in that stripper anodes are larger and that 31 pairs of electrodes per tank are used.

A typical tank house circuit may, for example, contain 10 commercial and two stripper sections. Depending on electrode current density at which it is desired to operate to fit production schedules (varying from 10 to 16.6 amp per sq ft), circuit current will vary from 7220 to 12,000 amp.

At 11,000 amp, potential across a commercial tank is approximately 0.21 v, across a 38 tank section approximately 8 v, and across a 12 section circuit approximately 96 v. Supply voltage is necessarily somewhat higher to take care of resistance drops through connecting bus bars and contacts. Because of the fact that both electrodes are copper, there is

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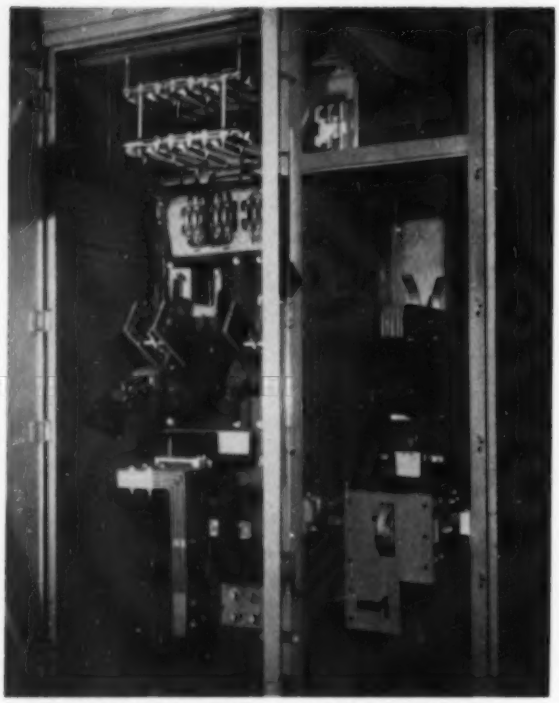


At left of this dc output breaker for the mechanical rectifier are two manually operated negative disconnects. Two magnetic dc positive breakers are on the right.

no open circuit voltage or back emf developed by the tanks.

In tank house operation it is necessary to prevent too great a concentration of dissolved copper in the electrolyte. This is accomplished by the insertion in one of the circuits of a *liberator* section, differing as far as the electrical features are concerned from a commercial section in that the anodes are lead. Thus copper supplied by the solution and deposited at the cathode is not renewed at the anode, resulting in depletion of copper in solution.

Another typical tank house circuit may, for example, contain a 24 tank liberator section in series with 12 commercial sections. A liberator section must be handled with circumspection because when it is cut into the circuit it is electrochemically a battery under charge (lead-copper electrodes, sulphuric acid-copper sulphate electrolyte), emitting oxygen and hydrogen at anode and cathode respectively. These gases are an explosive combination under the right conditions. For this reason a liberator section is provided with good ventilation, and smoking in the area is prohibited. It is also customary to guard against striking electrical arcs by insulating cathodes from the connecting bus by wood blocking when cutting the section in or out of circuit, using short circuiting locks.

Open circuit voltage or back emf is about 1.35 v per tank or 32 v per 24 tank section. Charging voltage (voltage drop across section when cut into working circuit carrying 11,000 amp) is approximately 60 v. This includes internal resistance drop through tanks, and resistance drop across contacts and bus bars. Total supply voltage required for the circuit illustrated is 96 + 60, or 156 v.

In copper powder deposition the desired end product is not an electroplated sheet of refined copper but refined copper in granular or powder form. Tanks designed for this purpose use electrode pairs in which the anode is a sheet of refined *cathode copper* (product of commercial section) and the cathode is a sheet of lead. Copper deposition is in a granular form adhering loosely to the lead cathode from where it is recovered for further processing by periodically scraping it off onto the tank bottom using wood tools.

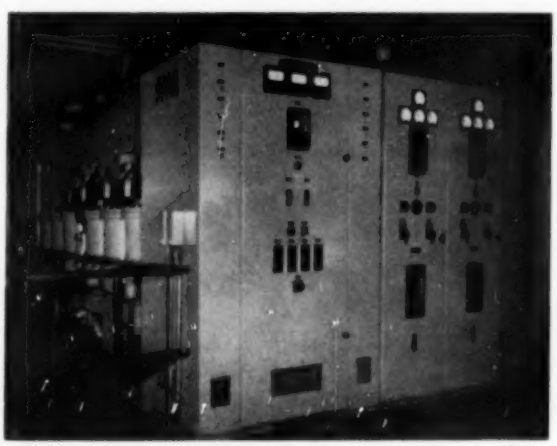
A newly charged copper powder section with clean electrodes develops an open circuit voltage at the instant of starting but of opposite polarity to that of a liberator section since the lead-copper electrodes are reversed in position. This voltage acts to boost rather than buck the supply voltage, but disappears as soon as the lead cathode has become plated with copper. From that point on the section has the same electrical aspect as a commercial section. However, the internal resistance is considerably higher than that of a commercial section because: A—A copper powder tank contains only 18 electrode pairs; B—surface area of an electrode is one third less than that of a commercial electrode; and C—electrolyte is of lower concentration than that used in commercial sections.

At 11,000 amp potential across a copper powder tank is approximately 1.40 v, across a 16 tank section approximately 22 v, and across a 4 section circuit approximately 88 v. Supply voltage is somewhat higher to take care of resistance drops through bus bars and contacts.

Power Supply Equipment

Several types of equipment are used at Carteret for supplying direct current to tank house circuits at variable voltage between 60 and 150 v. These are: A—Conventional motor-generator sets, B—steam turbine driven set, and C—rectifier sets.

Two conventional motor-generator sets manufactured by General Electric Co. are in use. Each consists of a single shaft, 3 bearing assembly comprised of a 2600 hp, 514 rpm synchronous motor driving two 900 kw multipolar dc shunt generators, and one 20 kw and one 10 kw end driven dc exciter for motor and generators respectively. Amplidyne generator automatic current control for each set is pro-



This is one of two 1500 kw, 250 v, dc output ignition rectifiers. Six of the twelve mercury tubes on the machine are at the left. Cubicles in the foreground contain auxiliary controls, and main operating controls and indicating instruments for both machines.

vided accurate to ±2 pct of maximum current of 12,000 amp (±240 amp). For power factor correction it is possible to operate driving motors to draw not less than 800 kva leading reactive, with an overall efficiency drop of 1/3 to 1 pct.

One steam turbine driven generator set manufactured by Westinghouse is in use. This consists of a 5200 hp, 3600 rpm combination impulse-reaction turbine driving two 1750 kw multipolar dc shunt generators at 360 rpm through twin herringbone reduction gears. Prime mover operates at 145 psi and 400°F total temperature, and exhausts into a 5400 sq ft surface condenser operating at 28.5 in. Hg vacuum or better. Steam extraction at 5 psi is possible in amounts as follows: 0 at 2400 kw load with 41,000 lb per hr throttle flow, 5000 lb per hr at 2450 kw load with 47,000 lb per hr throttle flow, and 20,000 lb per hr maximum at 2900 kw load with 62,000 lb per hr throttle flow.

The extraction feature is particularly useful in this application. Low pressure steam extracted is delivered to heat exchangers for maintaining tank house electrolyte at 150°F. Condensate from these heat exchangers is used to wash cathodes coming off the sections. Generators are rated 12,000 amp each at 146 v and are electrically independent. Each generator carries one tank house circuit.

Rectifier Sets

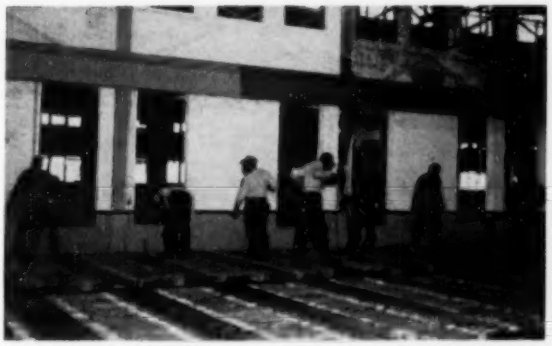
Rectifiers are fundamentally capable of high efficiencies since they merely switch energy and do not convert it from electrical to mechanical form and back again as do MG sets. Thus the only losses are incurred in contacts and auxiliary control devices.

Ignitron (mercury arc) rectifiers are classed as synchronous electronic switches and exhibit high efficiencies at voltages above 250 v dc. At voltages much lower than this the arc drop between electrodes causes their efficiencies to compare unfavorably with those of MG sets. Thus while ignitrons are used at Carteret for 250 v dc lighting and power service, their use is not economical on 60 to 150 v electrolytic circuits.

Mechanical type rectifiers, classed as synchronous mechanical switches exhibit high efficiencies at voltages above 50 v dc since the effect of contact drop is much less on this type of equipment. An experimental installation for electrolytic service has been made at Carteret of a mechanical rectifier unit manufactured by the I-T-E Circuit Breaker Co. of Philadelphia. The idea of mechanical rectification of high current at low voltage on a practicable basis was the subject of German patents acquired by the Government at the conclusion of World War II. I-T-E was asked by the Navy Dept. to work on a practical application in 1946 and since that date has developed this design for industrial use.

It is understood that General Electric Co. is also in position to furnish this type of equipment on domestic applications. It is also understood that an I-T-E installation at Buffalo Electro Chemical Co., installed in 1950 on 25 cycle input, has been operating successfully to date. The installation at Carteret, operating on 60 cycles, is more critical because of the necessary higher speed of moving mechanical parts and for this reason is still considered in the experimental category. However, the problems involved are not electrical but purely mechanical, and it is believed that they are capable of being resolved.

The basic mechanical element of the rectifier is a six phase synchronous motor driven mechanical commutator. The commutating device is not a rotating group of circular segments such as is commonly thought of in connection with a dc motor or rotary converter but is a battery of *poppet valve* type contacts operated by push rods. These push rods are operated in sequence by means of tappets and eccentrics from a motor driven cam shaft. In this respect the operation is mechanically comparable to that of an automobile engine valve, tappet, and cam shaft arrangement.



Workmen at the Carteret, N. J. plant of U. S. Metals Refining Co. straddle electrolytic tanks while taking care of details connected with the copper refinery tank house.

A pair of contacts is required for each phase of the six phase system, 6 pairs or 12 contacts in all. One contact of each pair connects its respective phase to the positive dc bus, the other connects it to the negative dc bus. Cam shaft lobe design and speed is synchronized with the ac supply so that each phase is connected to the positive bus during its positive half cycle and to the negative bus during its negative half cycle. With a six phase supply successive phases are connected to the positive dc bus every 60 electrical degrees as they go positive, and to the negative dc bus every 60 electrical degrees as they go negative. The resulting dc voltage and current waves at the load terminals have a 12 phase ripple, which is further smoothed out by the inductance of the system, so that the resultant dc wave form is substantially smooth.

Of course the practical operation of a mechanical rectifier at 12,000 amp dc is not as simple as applying ac voltage to a mechanism such as that described above and taking off direct current. The normal sinusoidal alternating current wave passes through zero from positive to negative and vice versa in zero time. Unless it is modified to remain at zero value for a small but finite time interval sufficient to allow the necessary overlap for practical operation of the commutation process, destructive arcs will form and the contacts will burn and freeze. The auxiliary means provided in the rectifier control circuit to accomplish this modification is the basic electrical feature of this equipment, and is that feature which makes operation possible at all.

Briefly, the flat spot in the ac wave as it goes through zero is attained by the introduction of harmonics produced by saturable reactors, called *pre-excitation* reactors, in the primary circuit. These reactors are included as a part of the primary supply transformer and are mounted in the same tank with the main power windings. Thus the supply transformer is an integral part of the rectifier equipment, and is connected on the secondary side to the commutator assembly by short, heavy bus bars.

Operating Experience With A Two-Foot Diameter Blast Furnace

by Russell C. Buehl

AN experimental blast furnace at the Bureau of Mines, Pittsburgh, was operated from September 1953 through January 1954 as a cooperative project with the U. S. Steel Corp. to determine the suitability of a small experimental furnace for studying the smelting of iron ores and agglomerated concentrates. This furnace shown in Fig. 1 was constructed by the Bureau in 1950 for the development of a process to recover manganese from open hearth slag and manganiferous iron ores.[1] In this previous work on manganese recovery, the furnace had been used successfully in the production of spiegeleisen from open hearth slag and manganese bearing ore from Aroostook County, Me., but the smelting of iron ores had not been attempted.

Investigation of the smelting of concentrated ore agglomerates is of significance at present, because the steel industry in the U. S., in future years, may be called upon to use increasing amounts of iron-bearing concentrates prepared from taconites or low grade ores as a replacement for present Mesabi ores. The Bureau of Mines conducts such research because of its concern with the supply of minerals required by industry. Work of such magnitude could not under present appropriations have been undertaken by the Bureau without assistance. The U. S. Steel Corp. supplied the raw materials and a large portion of the operating personnel for the tests.

Furnace Features

The furnace for which a vertical section is shown in Fig. 2 is not a scaled-down model of a large furnace, but rather it was designed for efficient operation as a small unit. The diameter inside refractories of the hearth and of the bosh at the tuyere level is 2 ft 2 in. The height from the tuyere to the stock line is only 10 ft—truly a low-shaft furnace. The bosh and lower half of the furnace stack is lined with carbon brick which has proven satisfactory.

The heat losses are disproportionately great when compared to a commercial furnace because of relatively large surface area to volume ratio which must occur in small equipment. Heat losses from the bosh, tuyere, and hearth region were about 3,600,000 Btu per ton of metal as compared to 300,000 Btu per ton reported for a 20-ft diam furnace.[2] From the standpoint of thermal balance in the highest temperature zone of the blast furnace, the small blast furnace theoretically should be able to obtain results comparable to a large furnace if the additional heat loss is made up by an increase in blast temperature of about 700°F.

The high blast temperatures required to offset the heat losses of the small blast furnace were made possible by the use of stoves employing pure alumina pebbles about 1 in. in diam as the heat transfer medium.[3] Water leakage from the hot-blast valves caused some delay of operations in the early tests, but the difficulty was corrected by replacing silver solder joints with welds, after which the auxiliary equipment functioned very well.

Another novel feature of the experimental blast furnace was the removable hearth, which was so constructed that it could be lowered, rolled out of place, the furnace cleaned, and another hearth replaced in about a day if the furnace was not badly scabbed.

Operation on Mesabi Ore

The furnace was blown in as a cupola in order to heat the hearth rapidly and avoid the buildup of solidified slag and metal in the hearth at the beginning of the test campaign. Coke was charged first until it was level with the top of the bosh 2½ ft above the tuyere. The next few charges consisted of six parts pig iron to one part coke with limestone and dolomite as fluxes. Successive charges

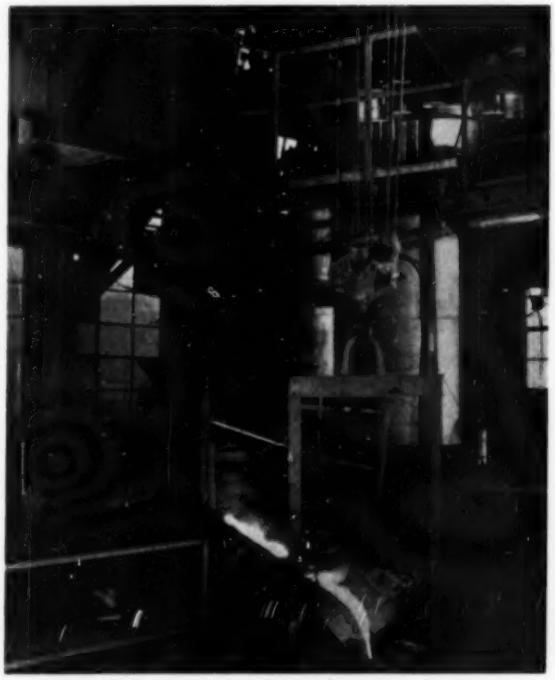


Fig. 1—The experimental blast furnace was constructed by the Bureau of Mines, Pittsburgh in 1950. Prior to the investigation of smelting iron ore agglomerates, it was used to develop a method of manganese recovery from open hearth slag and manganiferous iron ores.

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contained decreasing amounts of pig iron and increasing amounts of ore so proportioned that the last charge used to fill the furnace prior to blowing-in would contain no pig iron.

The blowing rate when starting as a cupola was that planned for the entire campaign, usually around 400 cfm. The blast temperature was 1200°F during cupola operation and increased over a period of a few hours to the temperature calculated for the initial ore burden. With this procedure, that portion of the charge containing no scrap reached the tuyere within 3 hr after the start of blowing. The burden and blast temperature were usually adjusted to those desired for the test within the next 24 hr. Satisfactory operating conditions yielding significant data were reached usually within 36 hr after *blow-in*. This starting procedure is considered very satisfactory for bringing a small furnace to stable operating conditions in a minimum time.

The initial tests were made on an ore shipment especially selected as representative of Group 13 ore as mined from the Mesabi Range deposits by the U. S. Steel Corp. In the first successful campaign, the furnace was operated continuously for 3 weeks on this ore. During the first two weeks of operation, the coke rate averaged about 2900 lb per net ton at blast temperatures of 1500° to 1600°F. The blast temperature was increased gradually to 1800°F which permitted a considerable decrease in the coke rate.

At the lower blast temperatures, the silicon contents of all but a few of the casts were within the range 0.5 to 1.4 pct with a weighted average of 0.94 pct. In the latter part of the test, the casts were mainly within the range 0.6 to 1.1 pct Si, with a weighted average of 0.76 pct. The sulphur contents of the metal, which were 0.3 to 0.4 pct with acid slags, decreased to 0.03 to 0.07 pct when slag of the normal composition was used.

In a repeat test on Group 13 ore, the coke rate was 2400 lb per net ton at a blast temperature of 1800°F. In the early part of the test, the silicon content was in the range 0.4 to 1.0 pct, but increased to 0.9 to 1.4 pct in the latter part with a weighted average of 0.72 pct for the entire test. This test was discontinued after 9 days of operation and an examination of the contents of the furnace during cleanout showed that there was no building up of accretions on the furnace walls.

Taconite Pellets Cause Difficulty

Tests were next conducted on pellets prepared from concentrated taconite ore. These pellets, which were mainly ¾ to 1½ in. in diam, had been agglomerated from magnetically concentrated taconite. The pellets consisted of an outside layer about ⅛ in. in thickness which was essentially hematite and an inner core of dense magnetite. The pellets were put over a ¼-in. screen prior to charging the blast furnace to remove disintegrated pellets.

When these pellets were smelted in the experimental blast furnace, an even descent of the charge could not be maintained for more than a few days. Inspection of the furnace during cleanout revealed massive scabs composed mainly of metallic iron. Part of this metal was in the form of hollow spheres. The layer of hematite on the surface of the pellets had been reduced to metallic iron in the shaft of the furnace while the magnetite core was reduced only as far as wüstite (FeO).

When the pellets reached the melting temperature of wüstite, which is 300°F lower than that of metallic iron, the wüstite flowed out leaving the metallic shell. This may well be the reason for the formation of metallic accretions when using these pellets and, if so, the rate of heating in the furnace might be of considerable importance.

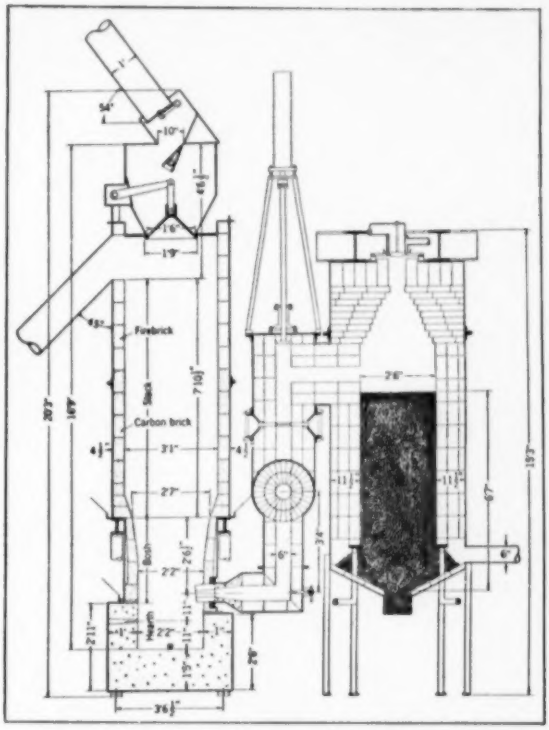

Fig. 2—Vertical section of experimental blast furnace shows that the furnace is not a scaled down model of a large furnace but was designed specifically for efficient operation as a small unit.

Material passed from the charge line of the small blast furnace to the melting zone in about 3 hr when there were no accretions and much faster after accretions formed. The longer retention time and slower rate of heating of materials in a commercial blast furnace should result in a greater reduction of the pellets prior to reaching the fusion temperature of wüstite. Furthermore, in a large furnace with tapered sides, any metallic accretions that formed might move through the furnace and not interfere with operations as occurred with the small cylindrical-shaped furnace.

The difficulties that were encountered with metallic accretion formation, therefore, might not occur when the same type of pellets were used in a commercial furnace. It was concluded that a furnace of greater height and somewhat large diameter with tapered sides probably would give better results with the ¾ to 1½-in. pellets.

The short tests on these pellets indicated that a coke rate comparable to that for reduction of Mesabi ore might be obtained at lower blast temperatures. The formation of metallic accretions prevented determination of either the maximum blast temperature or maximum blowing rate that could be used with the pellets.

Other ores which also were tried in the experimental blast furnace included taconite pellets of smaller size than those mentioned above, sinter, and agglomerated Cuban laterite ore. Work on these raw materials was restricted to 4 to 5 day campaigns.

The smaller pellets, mainly in the size range 5/16 to 9/16 in. with a smaller percentage as large as ¾ in., were supplied by the U. S. Steel Corp. from experimental runs. These pellets were passed over a 3/16-in. screen prior to use in the blast furnace to remove fine ore abraded from the pellets. Blast temperatures of 1400° to 1550°F were employed during the first 3½ days of the test, but the hearth remained cold and the furnace was difficult to cast although the silicon content of the metal averaged over 1 pct.

Increase of the burden and of the blast temperature to 1950°F improved operations and appeared encouraging, but the test was discontinued after a few casts. Sulphur contents of most of the casts ranged from 0.06 to 0.08 pct. On removing the hearth at the end of the test, all of the charge dropped out, indicating that these pellets would be less inclined to form accretions than larger pellets in this small furnace.

Sinter and Cuban Laterite Tests

In the campaign employing standard blast furnace sinter, the furnace was run first on a mixture of 60 pct Mesabi ore and 40 pct sinter at a blast temperature of 1400°F. The proportion of sinter was then gradually increased to 100 pct. A moderate increase in blast temperature was found necessary as the proportion of sinter was increased. The test was terminated after four days of operation. No accretions were found in the furnace. During the last three days of operation, the silicon in the metal varied between 1.1 and 1.7 pct, and the sulphur between 0.016 and 0.04 pct.

The Cuban laterite ore contains both chromium and nickel, and though the silica content is quite low (5 pct) the alumina (11½ pct) is much higher than for normal blast furnace ore. This ore, which was purchased from the Bethlehem Steel Co., had been nodulized in a rotary kiln. The resulting slag contained from 17 to 20 pct silica, 30 to 33 pct alumina, 40 to 45 pct lime, and 6 to 10 pct magnesia.

Slags usually had a temperature of around 2900°F, and were quite fluid. Operations appeared to improve as blast temperatures and burden were increased. A maximum blast temperature of 2125°F was used with these ores during the last half day of operation.

Comparison of Ore, Taconite Pellets, and Sinter

The original plan of research was to compare furnace operation on taconite pellets and on Mesabi ore. However, formation of metallic accretions when using the ¾ to 1½-in. taconite pellets, which limited furnace operations to about five days, was much worse than for any of the other ores tested and prevented an accurate comparison of this nature.

The research indicated that considerable information on blast furnace smelting can be obtained from tests with a small furnace. Relative desirability of different ores and agglomerates can be evaluated with respect to various factors, provided these raw materials did not result in serious accretion formation in one week of operation. In such a comparison, most of the operating conditions must be the same.

The simplest comparison is probably the coke rate necessary to produce metal within a desired silicon range for different ores at the same blowing rate and blast temperature. Under such conditions, the ore most easily reduced in the stack by carbon monoxide would have the lowest coke rate, provided other factors such as the slag volume per ton were not greatly different. Such operations determine in effect the relative reducibility of various ores, but are more realistic than the usual laboratory tests run at a definite temperature. Distribution uniformity of ore or pellet charges in relation to coke which controls the amount of gas chaneling is another important factor affecting the coke rate, but which is not given proper consideration in the usual laboratory reducibility test.

Although the length of the runs was too short for an accurate determination of coke rate at any blast temperature for the raw materials used in the experimental blast furnace, with the exception of Group 13 Mesabi ore, a comparison of coke rate at various blast temperatures indicates that:

A—At blast temperatures of 1350° to 1600°F, the ¾ to 1½-in. pellets gave a lower coke rate than other ores at the same blast temperature and, therefore, appeared to be more easily reducible.

B—The coke rate for the 5/16 to 9/16-in. pellets was only slightly higher than that for the larger pellets and less than that for the Group 13 ore at the same blast temperature.

C—Coke rates for Group 13 ore and blast furnace sinter were virtually the same.

D—Nodulized Cuban laterite ore gave coke rates higher than the Group 13 ore at the same blast temperature.

Ore giving the lowest coke rate at a definite blast temperature is not necessarily the most desirable. An increase in blast temperature of a few hundred degrees decreases the coke rate considerably. Consequently, it would be more significant to compare ores at the maximum blast temperature at which smooth operations can be maintained. Such tests would require several weeks of operation for each ore tested and could not be undertaken during these experiments.

It was concluded also that an experimental furnace of greater height and somewhat larger diameter with proper batter and more insulation probably would give better results for the type of tests undertaken than the furnace used.

Acknowledgments

The program development and planning of most of the project was done by Max W. Lightner, manager, research and development, U. S. Steel Corp., and his associates, T. F. Reed, R. G. Thompson, and R. L. Stephenson. W. A. Knepper and J. J. Bosley, also of that corporation, participated in the supervision of the project. The program owes its success to the combined efforts of so many persons of the pyrometallurgy branch of the U. S. Bureau of Mines and cooperative employees furnished by the U. S. Steel Corp. that their contributions, which are greatly appreciated, cannot be mentioned individually.

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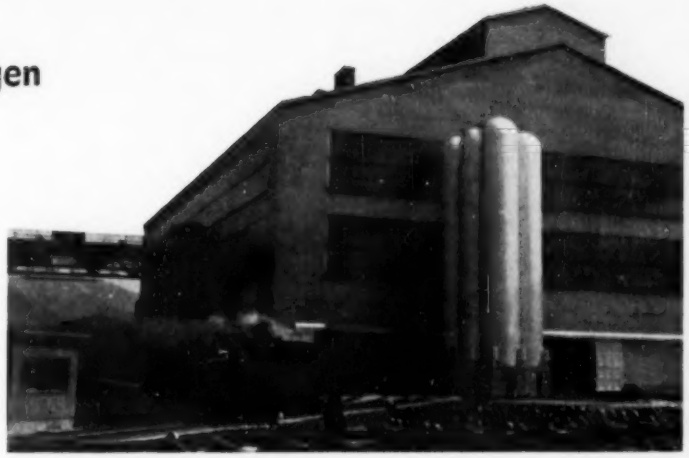
Development of Oxygen Steelmaking

by Otwin Cuscoleca



The OMAG oxygen steelmaking shop at Donawitz is housed in a building of special design consisting of charging, converter, and casting bays. This view of the shop shows the oxygen accumulators. For a section through the shop, and the layout for flow of materials, see Figs. 5 and 6.

The Austrian plants at Donawitz and Linz were the first to blow steel with high purity oxygen. The paper shows how the process was developed and gives a survey of the results achieved to date. Compared with open hearth steel of equal quality, the obvious advantages of oxygen steel are the low investment and conversion costs. The nitrogen content of oxygen steel is not higher than that of good open hearth steel. Rimmed and killed steel products made by the oxygen process are superior to rimmed and killed steel products made by the best open hearth process.

TWO new steel plants using the principle of making steel with high purity oxygen went into operation in Austria, one at Linz (Vereinigte Österreichische Stahlwerke) in November 1952 and the other at Donawitz (Österreichisch-Alpine Montangesellschaft) in May 1953. An estimated 330,000 metric tons were produced by the two plants in 1953, the first year of operation. In 1954 production by Austrian plants using the high purity oxygen approach will be raised to 450,000 metric tons. This will be in addition to facilities in Huckingen (Mannesmann A. G., Düsseldorf, Germany), those planned and under construction at Hamilton, Ontario, Canada (Dominion Foundries and Steel, Ltd.), and elsewhere. It is expected that by 1955 a total capacity of one million tons of high purity oxygen steel will be available.

At the close of the last century, because of increasing availability of scrap, open hearth steel production forged ahead at an ever increasing pace. This caused world steel production to rise faster than pig iron production and finally to overtake it. In 1911 65 million tons of steel were produced against 64 million tons of pig iron. World steel production was 209 million tons in 1951, while pig iron output was only 147 million tons. The 1951 breakdown for steel shows 118 million tons made from pig iron, while the balance, 91 million tons or 44 pct of the total tonnage of steel produced, was made from scrap. This increase in crude steel production on a basis of high scrap consumption led to the world market scrap shortage and soaring scrap prices. In the United States and other nations, production of converter steel has fallen far behind that of open hearth steel. The surplus of processing scrap, unabsorbable by converting mills and thus a welcome source for open hearth and electric melting shops in the past, together with regular scrap purchases, no longer can meet requirements of both open hearth and electric steel production in Europe. In addition, recent modification of the basic bessemer process, blowing with oxygenated blast, substantially raised the scrap capacity of these plants. Any further expansion of world steel production will necessitate utilization of self-containing processes in regard to scrap because of the shortage.

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Development, Especially in Europe, of Metallurgical Uses of Oxygen

There is nothing new in the idea of lowering the nitrogen ballast in the refining of steel. Henry Bessemer in 1856 touched upon this problem. Similarly, it has been known that, by lowering the nitrogen ballast in the converter blast, the nitrogen content of blown steels can be reduced. Practical use of oxygen could not be contemplated before 1928–1929 when large oxygen production plants based on the Linde-Fränkel process were developed and tonnage oxygen of sufficient purity could be produced economically.

First metallurgical applications were in the blast furnace to increase pig iron production and lower fuel consumption by oxygen enrichment of the blast (Oberhausen-Rheinland). Durrer turned experience gained in this work to good use in the development of the low-shaft oxygen furnace (Gerlafingen-Liége-Oberhausen), which shows promise of utilizing inferior grades of coal for pig iron production.

In steelmaking, it seemed logical to investigate the possibilities of oxygen enrichment of the blast in basic converting mills. Experiments were started at Max-Hütte (Germany) in 1924–25[1, 2] and ended in completion of the process which, a few years ago, was adopted for full scale production in the Ruhr and in Belgium.[3-6] A number of other experiments aimed at elimination of nitrogen by using mixtures of oxygen and steam and of oxygen and carbon dioxide to achieve a better quality product.[7-9] Apart from obtaining a better quality basic bessemer product and permitting a higher percentage of scrap in the charge, these experiments resulted in a reduced blowing time. Several million tons of steel a year are being produced in Europe by this process.

In open hearth steelmaking, first attempts to increase production through the new tool oxygen were made in the United States, both by injecting it into the bath by means of a lance and by using it for the melt down through the end burners. Production increases achieved were reported to be from 10 to 15 pct. However, it was impossible, to the knowledge of Austrian steelmakers, to find a satisfactory solution to the problem of reduced roof life.

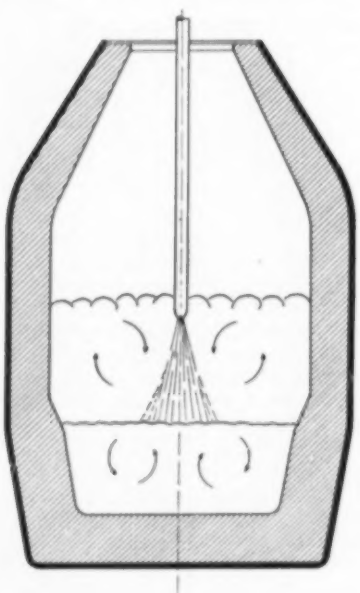


Fig. 1—Converter of symmetrical design as used in the oxygen steelmaking shop, Donawitz.

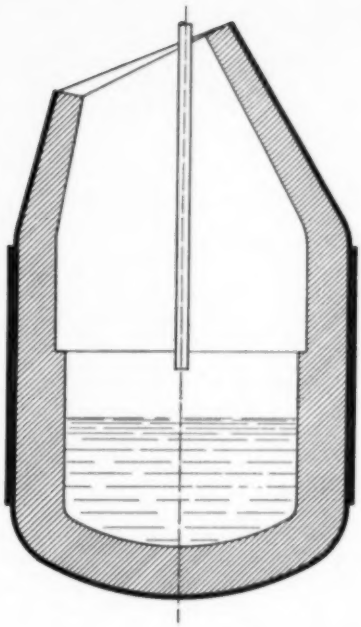


Fig. 2—Converter of unsymmetrical design as used in the oxygen steelmaking shop, Linz.

Oxygen is used in electric steelmaking, too, in the production of low carbon stainless steel and for a general production increase at the cost of reduced roof life.

The Development of Oxygen Steelmaking

While most open hearth shops could keep hot metal in the charge as low as 20 to 40 pct and were only recently forced by acute scrap shortage to raise hot metal to 50 and 60 pct for a very long time, the Donawitz open hearth shop was less fortunate. Hot metal charges of 70 pct and over had to be used, which is near the economic limit of this process. Any increase in production would depend on the amount of circulating scrap available in the home plant. This tonnage of scrap is too small for the open hearth shop to operate on an economically favorable basis.

The theoretical and practical work from which the new oxygen steelmaking process emerged dates back several years. In 1941 experiments were run at August Thyssen-Hütte and at Hagendingen to find out whether the Donawitz-type pig iron lent itself to blowing in a basic converter with normal air blast. The average analysis was: 3.7 to 4.3 pct C, 0.5 to 0.85 pct Si, 2.5 to 3.2 pct Mn, 0.10 to 0.25 pct P, and 0.02 to 0.04 pct S. Results showed that while heats could be blown under certain conditions (silicon content raised above approximately 0.6 pct minimum and elevated hot metal temperatures), the heat balance for normal blows would be on the low side. Furthermore, the yield was low because of accretions and heavy slopping, and the blown steel was inferior in quality to open hearth steel as a result of nitrogen pickup during the blow. Experiments continued after the war were discouraging. At the same time experiments had been started with pure oxygen as the refining medium[11-14] and, after preliminary theoretical work, it was decided to tackle the problem by using high purity oxygen. Experiments were started in 1949 at Donawitz by Rösner and Kühnelt and at Linz by Trenkler and Hauttmann.[15-21] Both groups used makeshift equipment: a 5 ton and later a 10 ton vessel at Donawitz, and a 2 ton and then a 15 ton vessel at Linz. Within

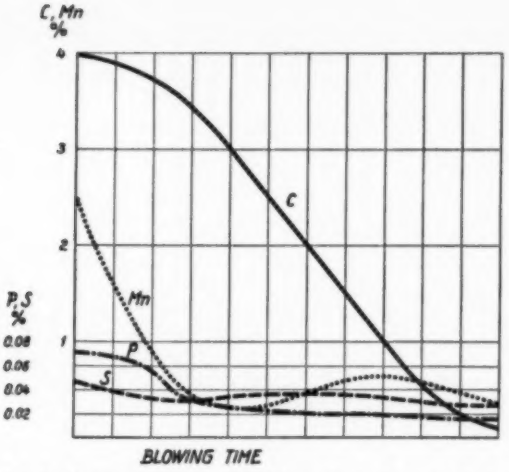


Fig. 3—Progress of process in making a heat of oxygen steel.

a short period, these experimental plants yielded results that were so favorable[23, 25] that final decisions could be made in 1950 and 1951 by the two companies, and the erection of production units started at Linz and Donawitz. It should be mentioned that the promising results from experimental and pilot operations were realized on a production basis after a few weeks and, in time, surpassed.

For a better understanding of the reasons that led to this special approach, these facts should be borne in mind:

1—In regard to conversion costs, efficiency, and plant costs, the converting processes are superior to the open hearth process with hot metal as the primary charging material.

2—It is mainly the high nitrogen content that makes converter steels inferior in quality to open hearth steels and suitable only for a restricted range of uses.

3—Adaptation of the conventional converting process is possible only with specific pig iron analysis, frequently difficult to obtain, if at all, from a given supply of pig iron.

The application of high purity oxygen as the refining agent combines the qualitative advantages of open hearth steel with the merits of the converting process as to productivity.

From earlier experimental work[14, 24] it was known that bottom-blown converters were unsuitable for working the blow with high purity oxygen. The high temperatures caused rapid destruction of the tuyere bottom. Therefore, oxygen must be blown from above against the surface of the bath. Experiments showed conclusively that this method was capable of producing a steel of excellent quality, provided certain conditions were maintained.

By suitably spacing the nozzle of the jet device with respect to the surface of the bath and by appropriate adjustment of gas pressure at the nozzle, the oxygen jet can be made to impinge on a relatively large area of the bath surface. When all adjustments are correct, the oxygen jet will be turned into a turbulent stream, equalizing to a certain extent the difference in pressure between center and periphery of the impingement area. Impinging on the surface of the liquid bath, the oxygen mols instantaneously start reactions leading to the formation of FeO, part of which goes into the slag, while part enters the bath by diffusion. Further, CO is evolved in the mechanism which gives rise to a vigorous boil. In the reaction zone, enlarged by ejected splashes and droplets of metal, temperatures close to the boiling point of iron prevail. Temperatures of this order promote solubility of the oxygen in FeO and Fe, and increase the diffusion rate of FeO through the metal bath. The evolution of CO is strongest in the immediate vicinity of the reaction zone. This causes an agitation of the bath which carries a continuous stream of metal to the reaction center where it is oxidized. The carbon boil also results in a mechanical mixing of the FeO with the bath, whereby the oxidizing process is propagated throughout the contents of the converter. With this reaction mechanism, a slag initially high in FeO is formed. Together with splashing metal particles and bubbles of CO, this slag forms a foaming iron-slag emulsion covering the metal bath. Extremely high temperatures cause the initially high FeO content of the slag to diminish rapidly by reaction with the carbon. In good practice the risk of heats becoming overoxidized is negligible. Highest temperatures occurring in this exothermic chain of reactions are within the area exposed to the action of the oxygen jet, i.e., on the surface of the bath, which results in a highly effective heating of the bath from above. Thus, fundamentally, conditions in the oxygen steelmaking process resemble those of open hearth steelmaking. They are even better in that the process progresses much more rapidly because of high temperature of the bath and slag in the primary reaction area and also because of the vigorous boiling action. Fig. 1 shows the progress of reactions schematically in the symmetrical converter developed for the Donawitz plant. As will be seen from Fig. 2, the conventional converter design was followed closely in the construction of the Linz vessel.

Fig. 3 shows a time study of the oxygen refining process on a basis of Donawitz-type pig iron containing about 4.0 pct C, 2.5 pct Mn, 0.2 pct Si, 0.075 pct P, and 0.05 pct S. It will be noted that with suitable control of the blow it is possible to complete dephosphorization prior to decarburization. This not only holds true for low phosphorus steelmaking iron, but also for pig iron higher both in phosphorus and in silicon.[26, 27]

Large Scale Operation

Plant: Based on process fundamentals, new steelmaking shops have been erected at Linz and at Donawitz which are producing 1000 metric tons per

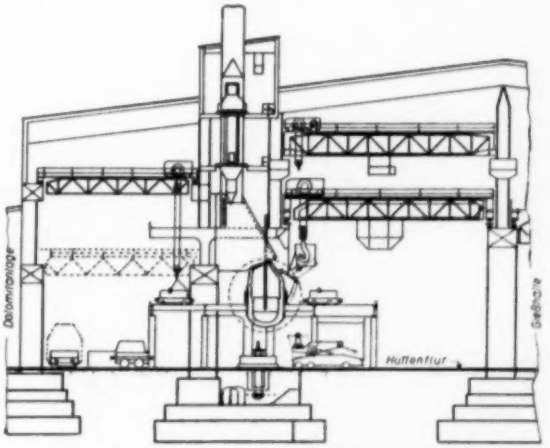


Fig. 4—Section through the oxygen steelmaking shop, Linz.

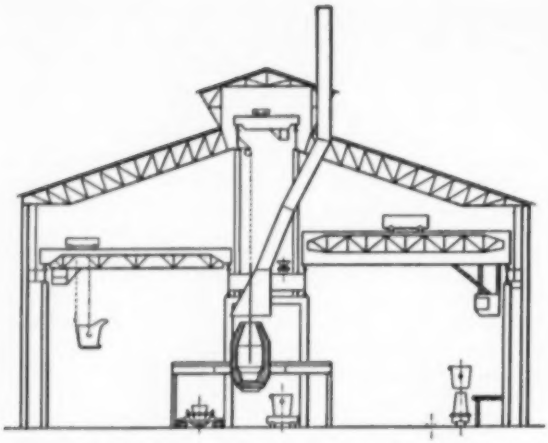


Fig. 5—Section through the oxygen steelmaking shop, Donawitz.

24 hr. Fig. 4 shows a cross-sectional view of the oxygen steelmaking shop at Linz, erected in line with the existing open hearth shop. The ÖAMG oxygen steelmaking shop at Donawitz is accommodated in a special building consisting of charging bay, converter bay, and casting bay, shown in Fig. 5. This arrangement is favorable for material flow as illustrated in Fig. 6; some stages of the operation are shown in Fig. 7.

With an overall length of 265 ft, the 3 bay building is 180 ft wide and contains, in this first stage of the project, one 500 ton hot metal mixer and two converters, each with a charging capacity of up to 33 metric tons. In planning the building, provision was made for the erection of a second mixer and a third converter, which would double production.

The numbers in the following paragraphs refer to the numbered parts of Fig. 6. From the blast furnace the molten iron is conveyed in ladle cars (3) to the oxygen melting shop and stored there in the hot metal mixer (1). (See Fig. 7a.) The mixer pours into transfer ladles positioned on a weigh-bridge (4) for positive adjustment of charging weights. The charging of hot metal from the transfer ladle is effected by tilting the converter (2) to the charging bay side. (See Fig. 7b.) Cooling scrap is placed on gravity chutes and charged into the converter from the charging bay side. Storage of lime is in high-line bins (11) positioned in the center bay and fed by means of drop-bottom buckets (10). From the bins a high-line skip car transfers the lime to inclined chutes (12) feeding directly into the converter when in the upright position.

The required volumes of oxygen are supplied from a tonnage oxygen plant rated at 70.630 cu ft of 98.5 to 99.5 pct O per hr. At a pressure of 356 psi, the oxygen is led to accumulaters (8) by an all-welded pipe line with short-length copper tubing incorporated for safety. In the feed line to the converters, pressure is kept at a constant level by Askania pressure regulators. A reducing valve actuated manually from the control panel on the converter platform permits positive adjustment of oxygen pressure required at the entrance to the nozzles. Connection with the retractable jet devices (9) is made by rubber hoses, and the same applies for water connections serving the cooling jackets of the lances. There are two lances, one a stand-by, for each converter. If necessary, lances can be changed in a few minutes. When the blow is finished, the converter is lowered to the pit side and the steel flows into a pouring ladle placed under the converter by a simple transport buggy. A crane (16) lifts the ladle (5) from the buggy, and the molten metal is poured into big-end-down molds positioned on casting buggies (7) for direct delivery to the stripping bay. (See Fig. 7d.) Removal of slag is on a track below the converters in the direction of the charging bay (14).

Two cranes of 65 ft span and each of 50 and 15 ton capacity are provided for charging bay operations (15) while the converter bay is served by a 10 ton high-line crane (17) of 21 ft span. The casting bay is covered by a crane of 75 and 15 ton capacity (16) and a 25 ton service crane (18) of 66 ft span.

Gases escaping from the converter mouth during the process are removed by hoods (19). Interference on crane runways was avoided by designing them to lead in an oblique angle from the center bay into the casting bay; at present they terminate in the open air about 145 ft above floor level.

In contrast to the conventional converting process and because of elimination of the nitrogen ballast, working of a heat with oxygen produces only a small volume of off-gas escaping from the converter at a high temperature. Fresh air intake through the stack provides for adequate cooling of the hot gases. Dust carry-over by the escaping gases amounts to

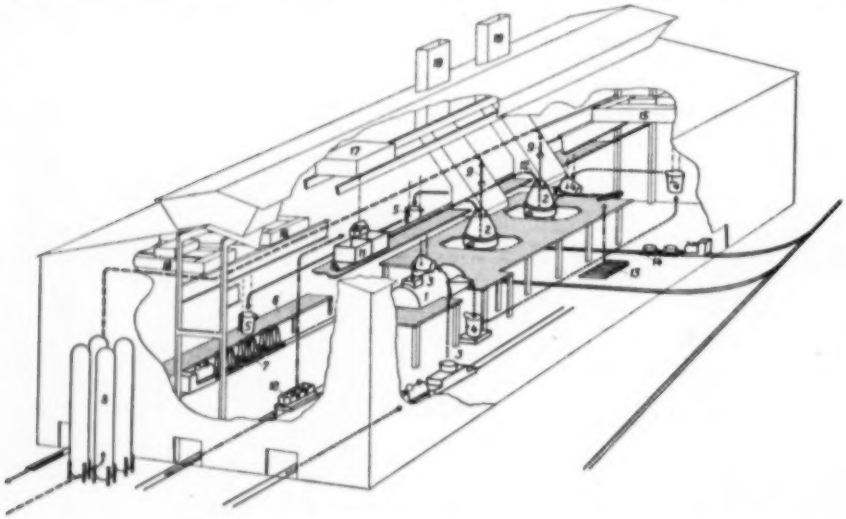
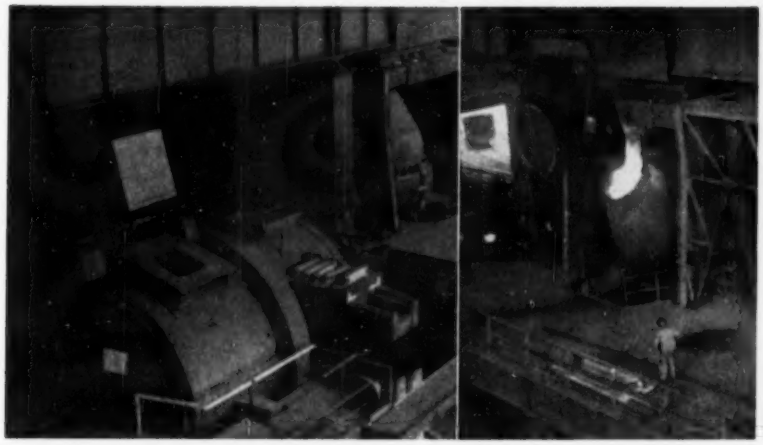


Fig. 6—Flow of materials in the Donawitz oxygen steelmaking shop.



a—Molten iron is conveyed in ladle cars from the blast furnace to the oxygen melting shop and stored there in the hot metal mixer. (See 1 and 3, Fig. 6.)

b—Charging of hot metal from the transfer ladle is effected by tilting the converter to the charging bay side. (See 2 and 4, Fig. 6.)

c—After charging, the converter is returned to the upright position while the blow is being made.

d—Molten metal being poured into big-end-down molds, which are positioned on casting buggies for direct delivery to the stripping bay. (See 5, 7, and 16, Fig. 6.)

Fig. 7—Some operations in the OMAG oxygen steelmaking shop, Donawitz.

4.5 to 5 oz per cu ft. Dust consists mainly of iron oxides. The iron loss caused by dust production is lower than 1 pct. This dust consists of infinitesimal particles. Determination of grain size has shown that 15 pct of the dust is more than 1 micron; dust particles up to 500 microns are rare exceptions. Sixty-five percent of the dust varies from 1 to 0.5 microns, while the balance is smaller than 0.5 micron. By and large this type of dust constitutes an aerosol, whose separation and collection by conventional methods entails certain difficulties. Experiments are being carried out at Linz and at Donawitz for an economic and efficient solution to this problem.

Investment Costs: Investment costs of the order of $5 million can be given for an oxygen steelmaking shop rated at 1000 metric tons daily production or a yearly average production of from 200,000 to 220,000 tons of steel. These costs are computed for the shop ready for operation and include all necessary special cars, but do not include outlay for the site, track layout, locomotives, and rolling stock. The estimate is based on European conditions. Additional cost to increase production from 220,000 to 440,000 tons will be of the order of $2.5 million. Thus, total expenditures for an oxygen steelmaking plant rated at a production of 440,000 tons per year will be approximately $7.5 million.

A comparison of cost sheets reveals that, based on European conditions, erection and installation of an open hearth shop of comparable capacity containing two 220 ton furnaces would involve a total expenditure of $9 million, while the costs for an open hearth shop with a yearly capacity of 440,000 tons, corresponding to the capacity of a 3 converter oxygen

shop, would run $16.5 million. The savings in capital investment and resulting saving in amortization per ton of steel produced will be of the order of 44 and 54 pct, respectively, compared with the open hearth installation.

Results Obtained from Current Operations: The production plant results surpassed those obtained in the pilot plant. The process is based on mixer metal of the following average analysis: 4 pct C, 2.5 pct Mn, 0.2 pct Si, 0.100 pct P, and 0.080 pct S. Calculated from the metallic charge (hot metal, plus scrap, plus metallic additions), the yield of sound ingots averaged 89 pct, 2 pct higher than originally estimated and on a level with yields obtained in open hearth shops.

On an average of 18 pct scrap in the charge, approximately the amount of scrap produced by the plant, the balance of materials used per ton of steel in lb per ton is given in Table I.

The volumes of slag carried vary from 11 to 12 pct. The CaO content of the slag varies between 35 and 40 pct; the average for Fe is approximately 14 pct while the Mn content is as high as 15 pct, caused by specific analysis of hot metal used. Slag of this composition can be used in the blast furnace charge and represents a potential credit item. The iron losses into the slag are, therefore, lower than under the pig iron-ore process.

Converters are lined with pressed brick made from a mixture of sintered magnesite and tar. Bricks are used in the unburnt state. Within a short period it was possible to increase the lining life to 200 to 250 heats. Based on this performance, the consumption of refractories is about 18 to 24 lb per ton of steel produced. The open hearth furnace requires 36 lb of refractory brick for the furnace lining with some 40 lb of sintered dolomite added for fettling operations for a total of 76 lb per ton of steel.

The average consumption of oxygen is 1940 cu lb per metric ton, including the volume required for the heating-up of converters after relining. An oxygen efficiency of 95 pct has been established. Oxygen jet devices have a life of several hundred heats and maintenance costs are low. Water requirements for cooling the lance amount to 2.64 gal per sec, lower than the cooling water requirements of an open hearth furnace.

From the angle of energy requirements, the favorable economics of heat in the oxygen converter brings about a savings of at least 75 pct compared with the open hearth process. A consumption of 1940 cu ft O per metric ton under the oxygen melting process, representing an energy value of 0.25×10^6 kg-cal max, stands against a heat input of 1×10^6 kg-cal for the open hearth furnace.

Table I. Balance of Materials Used Per Ton of Steel in Pounds Per Ton*

Materials	Lb per Ton
Pig iron	1884
Scrap	360
Metallic additions	3.2
Total	2247.2
Melting and other losses	213.2
Liquid steel	2034
Pit scrap	34
Ingot yield	2000

* On an average of 18 pct scrap in the charge, approximately the amount of scrap produced by the plant.

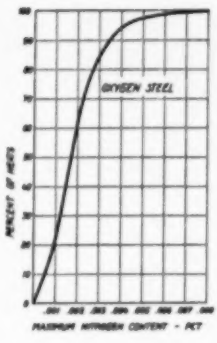


Fig. 8

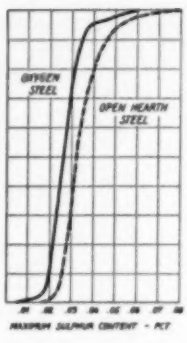


Fig. 9

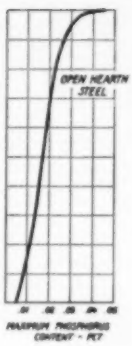


Fig. 10

Fig. 8—Accumulative frequency for nitrogen in the first 2000 heats of soft rimming oxygen steel produced at Donawitz.

Fig. 9—Accumulative frequency for sulphur in the first 2000 heats of soft rimming oxygen steel and in open hearth heats made at the same time from the identical grade of pig iron at Donawitz. No desulphurization in either case.

Fig. 10—Accumulative frequency for phosphorus in the first 2000 heats of soft rimming oxygen steel at Donawitz.

With approximately equal costs for materials handling, mixer, and pit operations, conversion costs for the oxygen steelmaking process are substantially lower than for the open hearth process. In the fourth month of operation, the conversion costs for the new oxygen melting shop dropped to 52 pct of those of the open hearth shop.

Briefly summarized, the economic advantages of the oxygen steelmaking process over the open hearth process,[10] realized after a short period of operation, are: 1—Capital expenditure and, with it, amortization per ton of steel are lower by 44 and 54 pct, respectively. 2—Energy requirements show a reduction of 75 pct. 3—The productivity per man-hr is higher than that obtainable in good open hearth practice. 4—There is a continuous flow of ingots into the blooming mill. A comparison of charging costs may be somewhat unfavorable with respect to lower percentages of hot metal in the open hearth charge. However, this drawback is offset by lower conversion costs.

Increased steel production cannot always be obtained by charging the open hearth furnace with higher percentages of scrap, since the limited availabilities of scrap necessitate use of higher hot metal charges. Therefore, it might appear quite useful to unburden the open hearth from high hot metal percentages in the charge by a partial change-over to the oxygen steelmaking process, as in Austria. Well balanced application of both open hearth and oxygen steelmaking presents a scheme by which, as a final combined result, a reduction of average costs for either process can be achieved. Hot metal essentially will be processed in oxygen converters under favorable conditions. Scrap will be allocated primarily for open hearth melting, thus turned to best advantage. Open hearth shops that are no longer burdened with high hot metal charges will be able to reduce charging costs and will achieve an increase in production at lower conversion costs.

The oxygen steelmaking process will be of particular interest where pig iron high in phosphorus

or silicon is produced, since refining an iron of this composition in the open hearth is bound to cause substantial difficulties and costs.

Quality of Products

From the nature of the charge and melting process, oxygen steelmaking incorporates all requirements favorable for production of steel particularly low in impurities and quality impairing elements.[81]

Nitrogen content of pig iron rarely rises above 0.010 pct.[80-85] Neither the technically pure oxygen used as the refining agent (containing a maximum of 0.3 pct N_2) nor scrap and other materials added carry substantial amounts of nitrogen. Thus, nitrogen pickup of the metal bath can be termed negligible despite high temperatures developed in the oxygen bath reaction area. Above all, refining reaction continues vigorously up to the end point and effectively causes elimination of such major amounts of nitrogen present in the bath because of untoward conditions. This set of conditions results in a low nitrogen content of the steel produced. Frequency curves prepared at Donawitz have shown an upper limit for nitrogen content of oxygen steel of 0.002 to 0.003 pct, with 95 pct of heats blown showing a maximum content of 0.004 pct N_2.

Fig. 8 presents the accumulative frequency curve for the nitrogen contents of the first 2000 heats of 30 tons each of soft rimming oxygen steel produced at Donawitz. On the whole, it can be said that oxygen steel is even lower in nitrogen than open hearth steel.

The effective heating of slag cover and metal bath from above and the boiling action setting in at an early stage of the process are highly beneficial. They promote the formation of a fluid and reactive slag relatively high in lime. This condition does much to speed up the rate of phosphorus and sulphur removal from the bath, leaving the steel low in both elements. Again, oxygen steelmaking is distinguished from conventional converter processes.

If a steelmaking iron of moderately high sulphur and phosphorus content is refined to carbon contents below 0.10 pct by the oxygen melting process, the final content of either element in the finished steel would be difficult to surpass in open hearth melting without additional steps. Fig. 9 shows accumulative frequency curves for sulphur content of 2000 heats of soft rimming oxygen steel blown at Donawitz from typical hot metal charges (average analysis 4 pct C, 2.5 pct Mn, 0.2 pct Si, 0.10 pct P, and 0.08 pct S) with lime additions of 4.5 pct, and for the sulphur content of open hearth steel produced at the same time from pig iron of identical chemistry. Fig. 10 shows the accumulative frequency curve for the phosphorus content of the same 2000 heats of open hearth steel.

If a pig iron lower in sulphur is available, or soda desulphurization in the ladle is practiced, the sulphur content of the finished oxygen steel can be kept at an even lower level.

It will be noted, Fig. 3, that during the last minutes of the blow, the P and S curves level out to such an extent that even structural steels caught at higher carbon contents are not noticeably higher in sulphur and phosphorus.

Experimental blows have been run on pig iron grades containing up to 1.5 pct P. Results revealed conclusively that rapid formation of a highly reactive slag enables the oxygen steelmaking process to keep the phosphorus content of the finished steel well below 0.05 pct,[86, 87] despite the high phosphorus in the charge. The same results were obtained by Springorum[25, 26] on blowing specific basic bessemer iron with about 2.0 pct P on the principles of oxygen steelmaking.

The vigorous boil, continuing right up to the end point, and the circulation of the bath metal, a distinguishing feature of the new process, effectively counteract overoxidation of the bath, providing the heat is properly worked. For this reason oxygen steels never suffer from the deteriorating effects of high FeO contents. Direct oxygen determination by the hot extraction method yielded oxygen contents of 0.040 pct max for heats that were decarburized to an end carbon of from 0.05 to 0.10 pct C. This result compares easily with the oxygen contents of open hearth steel in the same carbon range.

Since the oxygen steelmaking process is self-contained as to scrap, it is obvious that with a hot metal poor in detrimental elements, such as copper, the oxygen steel will be comparatively free from elements that in the case of open hearth steel are entrained by the scrap used, and that can have an undesirable effect.

Because of the high temperature and low slag volumes, the manganese content of a heat of oxygen steel is generally higher than from an open hearth heat melted from the same grade of iron under the pig iron-ore process and refined to the same end carbon. With low manganese hot metal this means a saving of ferromanganese, while with high manganese iron no additions of ferromanganese are necessary.[15-28] It will be noted from Fig. 3 that the manganese oxidation curve has a definite hump. This and slag analysis show that under the oxygen melting process the oxidation of carbon during the last third of the refining period causes reduction of iron and manganese into the bath, while any throwback of phosphorus can be avoided by proper slagging practice and temperature control. It is also possible to maintain constant conditions to keep the

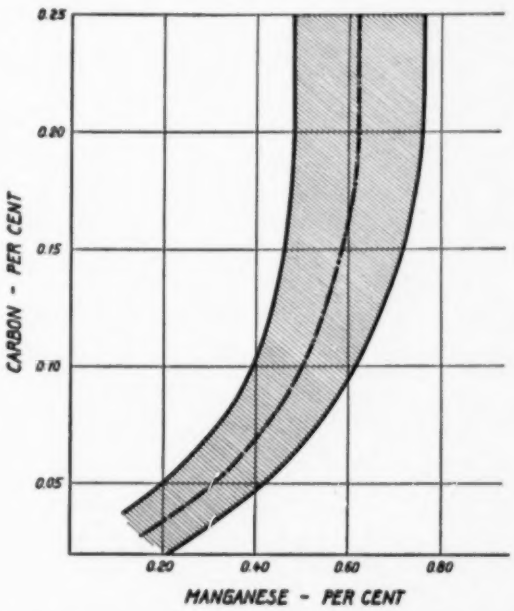


Fig. 11—Relationship of manganese and carbon contents based on analysis of heats made in the 30 ton converter at Donawitz. No recarburization.

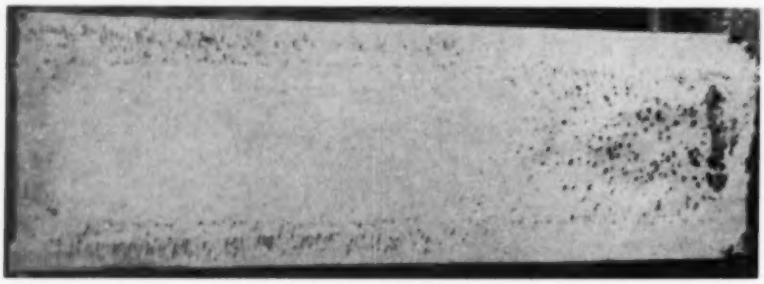


Fig. 12—Cross section of rimming, 4 ton oxygen steel ingot with 0.080 pct C and 0.45 pct Mn.

rise of the manganese hump within narrow limits, establishing a close interrelationship of the manganese and carbon contents with carbon as the governing element. This is demonstrated in Fig. 11 on a heat produced from the Donawitz-type of steelmaking iron of the usual chemistry. By adjustment of operating conditions, it is possible to obtain varying contents of manganese with given carbon contents, which, with a view to different end uses, offers a desirable amount of flexibility.

The particular purity of oxygen steel heats, in combination with easily maintained high tapping and casting temperatures, impart a high degree of fluidity to the heat. Thus, ingots cast from rimming heats have exceptionally few blowholes, segregates, and sonims.

Fig. 12 is a cross section of an ingot cast from soft rimming oxygen steel. Owing to the excellent uniform quality of this type of ingot, rejection of semi-finished bars high in inclusions has become rare.

The lower contents of phosphorus and sulphur and the reduced amount of segregation, particularly in regard to sulphur, over the better part of the ingot length will be noted from Fig. 13. This favorable condition enables Donawitz steelmakers to use 4 to 5 pct more of the ingot weight for qualities intended for deep drawing, or cold reducing, or similar applications in the case of oxygen steel, than in the open hearth process.

By the nature of the process, steels produced under the oxygen steelmaking method are distinguished by soundness, implying good workability as well as good mechanical properties of the finished product.[15-20]

A breakdown by uses of the 330,000 tons of oxygen steel produced in Austria in 1953 is given in Table II.

Experience gained from this varied manufacturing program is diverse and promising. Among other factors, the higher purity of oxygen steel results in a tensile strength in the as-rolled and normalized condition that is about 4270 psi lower than for Donawitz open hearth steel of comparable chemistry in regard to carbon and manganese.

From Fig. 14 it will be noted how far this condition affects the average chemical composition of a steel produced at Donawitz, both in the open hearth

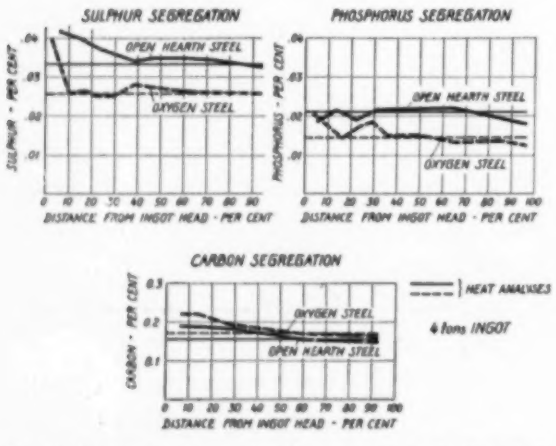


Fig. 13—Segregation of accompanying elements in rimming steels of 57,000 psi tensile strength made under both open hearth and oxygen steelmaking processes.

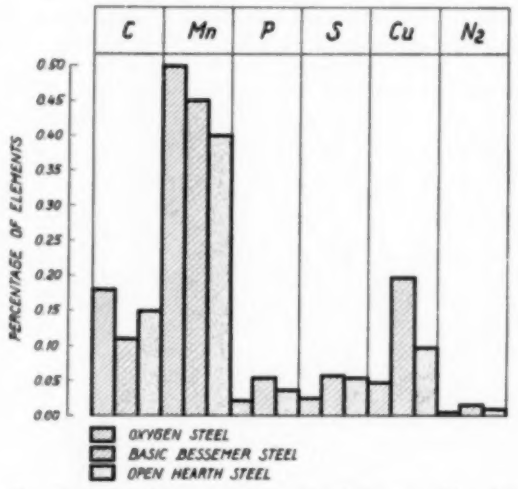


Fig. 14—Comparison of average chemical composition in steels of 52,000 to 64,000 psi tensile strength made under different processes.

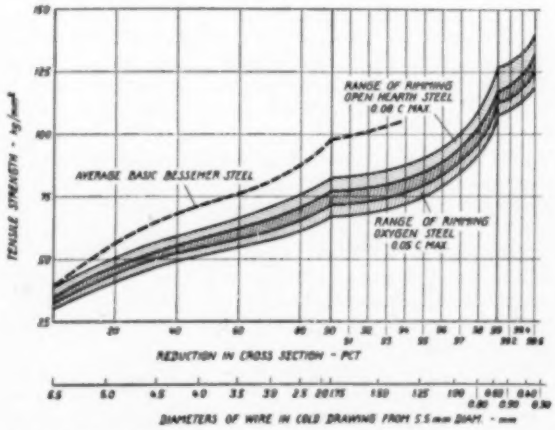

RANGE OF RIMMING OPEN HEARTH STEEL 0.08 C MAX
AVERAGE BASIC BESSEMER STEEL
RANGE OF RIMMING OXYGEN STEEL 0.05 C MAX
REDUCTION IN CROSS SECTION - PCT
DIAMETERS OF WIRE IN COLD DRAWING FROM 5.5 mm DIAM. - mm

Fig. 15—Strain hardening of wire drawn from rimming low carbon steels of different origins.

Table II. Breakdown of 330,000 Tons of Oxygen Steel Produced in Austria, 1953

Uses	Pct
Donawitz Production	
Structural, bar, and rail mill products	39
Wire manufacture	36
Welding electrodes	1
Strip and sheet, hot and cold rolled	12
Tube strip	5
Sundry uses	2
Linz Production	
Tin strip	26
Autobody sheets, cold rolled	11
Deep drawing sheets, hot and cold rolled	19
Galvanized sheets	6
Plates	10.5
Bar and section mill products	26
Commercial qualities	1.5

and in the oxygen converter—German standards St37 having a tensile strength of from 52,600 to 64,000 psi. The yield-tensile ratio is about the same for both steels made at Donawitz, while mean values obtained for elongation and reduction of area are slightly higher in the case of the oxygen steel.[35-38] Oxygen steels with carbon contents of 0.5 pct and less are free from red shortness and have excellent hot forming properties. This condition is most strikingly expressed by flawless surfaces and particularly sound edges of rolled products.

Oxygen steel has been used to great advantage in the manufacture of wire. Strain hardening values for oxygen steel have been shown to be at the lower limit of values obtained for comparable open hearth steels, and the susceptibility to strain hardening is perceptibly lower than in the basic bessemer steels shown in Fig. 15.

Dead soft rimming steels, widely used for wire manufacture, can be produced more economically in the oxygen converter than in the open hearth furnace. Soft rimming wire rods of 0.22 in. diameter rolled from oxygen steel stocks can be cold drawn down to 0.012 in. without any intermediate reheat. The processing of oxygen steel rods in various wire drawing shops has confirmed that even with higher carbon contents the cold forming properties of oxygen steel outmatched those observed in standard-grade open hearth steel, with the additional advantage that oxygen steel wire is less susceptible to cupping.

In the manufacture of welding electrodes from oxygen steel, it has been found that the wire fuses easily and smoothly when used in light and medium-coated electrodes. In gas welding oxygen steel rods give exemplary performances because of the absence of interfering slag films. In addition, oxygen steel electrodes are suitable for automatic welding where, with properly prepared surfaces, a good weld is obtained without spattering. In all the cases investigated so far, the beads were flawless.

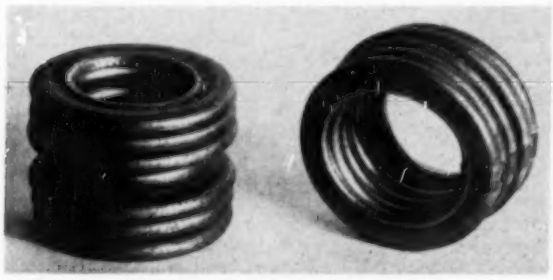


Fig. 16—Upsetting tests on electric-resistance welded pipes manufactured from 0.16 pct C oxygen steel.

There are undeniable advantages to the utilization of oxygen steel in the cold rolling of rimming low carbon strip. Less passes are required to reduce this material to the desired thickness than with open hearth steel. Consumers report less tendency toward the formation of ears in deep drawing than is observed when standard open hearth deep drawing sheet is used. This favorable behavior occurs because the oxygen steel is lower in impurities and segregation products. Since oxygen steel gives better performance than open hearth steel under conditions of pressure welding, oxygen steel is preferred for the manufacture of welded pipe from hot and cold rolled strip.

Evaluation of welding results shows a definite decrease in discards arising from unsound welds compared with open hearth welds. Where electric-resistance butt welding was concerned, higher speeds were possible, especially when dimensions were close to the limits of the process. The use of oxygen steel has proved to be of particular advantage in the manufacture of welded pipe that is subjected to bending operations subsequent to welding.

Fig. 16 shows a number of current upsetting test specimens from oxygen steel pipe. For the manufacture of precision steel tubing drawn to small diameters subsequent to mechanized gas welding, good deformability and surface finish (for nickel plating) of the weld are first considerations. Owing to the low contents of phosphorus and sulphur as well as the comparative freedom from segregates, no difficulties were encountered when welding tubes from slit oxygen steel strips. Again, this compares

Fig. 17—Parts manufactured from oxygen steel strip and sheet.

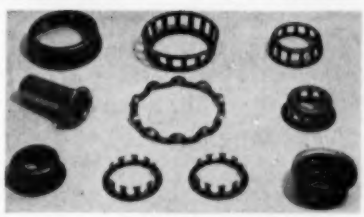


Fig. 18—Parts manufactured from oxygen steel strip and sheet.

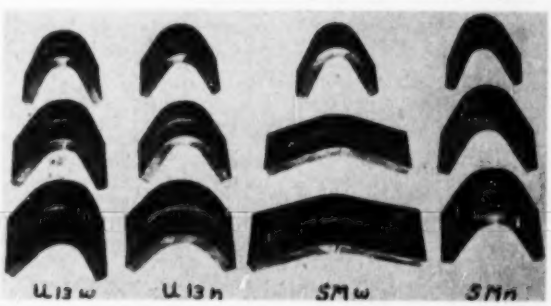


Fig. 19—Weld bending test specimens prepared from 1.57 in. plate. Comparison of specimens in the as-rolled, *w*, and normalized, *n*, condition cut from plates of rimming oxygen steel U13 and rimming open hearth steel SM with carbon contents of 0.13 and 0.10 pct, respectively. Specimens were cut from positions representing distances from the top of the slab of 10, 50, and 90 pct, respectively. Testing temperature, 68°F. (Data partly from Hauttmann.[38])

Table III. Values Obtained for Plate Rolled from Killed Oxygen Steel*

Heat Analysis	Sampling	Upper Yield Point, 1000 Psi	Lower Yield Point, 1000 Psi	Tensile Strength, 1000 Psi	Elongation ($L = 5.65\sqrt{F}$), Pct	Reduction of Area, Pct	Notch Impact Values** Ft-lb per Sq In.		Bending Angle of Weld Bend Test
							As-normalized	As-aged	
0.18 pct C 0.64 pct Mn 0.12 pct Si 0.04 pct P 0.013 pct S 0.057 pct Al 0.006 pct N_2	Top of 14 ton ingot as-normalized, longitudinal (1.575 in. plate)	46.2	43.1	69.4	26.4	61	750	n.d.†	>150°††
	Middle of 14 ton ingot as-normalized, transversal (1.575 in. plate)	44.2	41.2	67.5	26.4	63	746	433	>150°††
	Bottom of 14 ton ingot as-normalized	45.9	39.0	63.1	29.9	66	741	n.d.†	>150°††
0.20 pct C 1.49 pct Mn 0.47 pct Si 0.043 pct P 0.017 pct S 0.029 pct Al 0.007 pct N_2	3.15 in. plate, longitudinal, as-normalized	62.2	57.2	90.2	28.6	63	629	270	>120°††
	2.360 in. plate, longitudinal, as-normalized	61.2	57.7	89.3	30.0	66	731	443	>130°††
	1.575 in. plate, longitudinal, as-normalized	58.5	61.6	93.0	28.9	64	703	471	>150°††
	0.787 in. plate, longitudinal, as-normalized	60.6	59.2	86.6	30.1	63	774	513	>150°††
0.18 pct C 1.33 pct Mn 0.43 pct Si 0.037 pct P 0.022 pct S n.d.† Al n.d.† N_2	1.575 in. plate, longitudinal, as-normalized	57.9	n.d.†	84.5	28.6	63.1	760	564	158°††
	1.575 in. plate, transversal, as-normalized	59.6	n.d.†	86.5	26.4	60.2	615	447	162°††

* Partly according to Hauttmann.[28]
** DVMR, test piece.
† n.d.: not determined.
†† No fracture.

favorably with open hearth material, where troubles are not infrequent.

Its good cold forming properties and weldability have opened up a wide field of applications for oxygen steel in the form of thin sheets. At Donawitz and Linz, thin sheets were produced from oxygen steel. The total was 100,000 tons, the better part of which is accounted for by the Linz plant. Because of the favorable conditions under which rimming steels of particularly low carbon contents can be produced by the oxygen steelmaking process, the opportunity they offer for production of quality steel sheets and strip is obvious, especially for deep drawing applications. Results from Erichsen cupping tests are even better than those obtained for comparable open hearth steels. Various reports received note an exceptionally good performance of

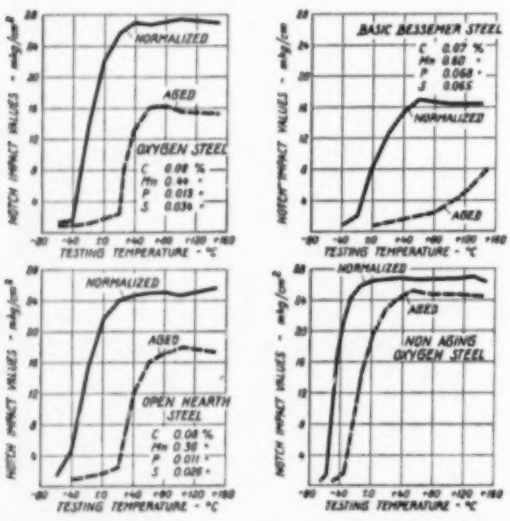


Fig. 20—Susceptibility to aging of steels originating from different processes.

oxygen steel sheets and strip and a decrease in discards. Part of these reports are included in recent publications.[16–18]

Figs. 17 and 18 show parts made from oxygen steel sheets and strip (cold rolled). Even with high degrees of cold work, no cracking was observed compared with the case of basic bessemer sheets.

In the field of steel structures, bridges, and shipbuilding, and boiler and container fabrication, plates made from rimming oxygen steel are equal or better than the qualities of open hearth products. It has been shown that oxygen steel plates are freer from brittle fracture than open hearth plates of identical strength. While results from weld bending tests have shown that brittle fracture is likely to occur in 0.59 in. St37 basic bessemer steel plates and in open hearth products with qualities of the same type from a thickness of 1.18 in. upward, this limiting thickness (according to Hauttmann[28]) is over 1.57 in. for St37 oxygen steel plates, as shown by Trenkler[27] in Fig. 19.

By deoxidation with silicon or other deoxidizers, it is possible to produce killed steels of excellent quality under the oxygen steelmaking process. Here again, the production of steel plate appears highly attractive. Plates used for welded constructions normally have carbon contents of less than about 0.25 pct, which can be obtained easily in oxygen steelmaking by catching the heat at that point or by subsequent recarburization. It has been shown that the aging and welding properties of killed oxygen steel together with freedom from brittle fracture, all-important for the performance of the steel in welded constructions, are better than those established for open hearth steel of identical strength.[22, 26, 27–29] It is possible to produce steels of higher strength and perfect freedom from brittle fracture by special deoxidation methods, as demonstrated by the experimental values listed in Table III (partly according to Hauttmann[28]), and by DVMR impact values in the transition range as shown graphically in Fig. 20.

The oxygen steelmaking process is not limited to

the production of low carbon steels, as evidenced by the 0.55C rail steel produced at Donawitz. It is impossible to present data from which conclusions may be drawn on service performance of these rails at present. The above-normal results obtained during inspection of the rails from laboratory tests, as well as fatigue testing and tests for wear resistance and welding properties, hold good promise that oxygen steel rails will stand up well in actual service.

Small-scale experiments have shown that the degree of purity displayed by oxygen steels easily meets the normal requirements for purity specified for case hardening and heat treating steels, and that oxygen steel is as responsive to heat treatment as is open hearth or electric steel.

The oxygen steelmaking process will be applied chiefly in the production of mass steels. However, in regard to quality considerations, the potential range of this process goes far beyond the production of commercial steels. In economy of operation and quality of product, the first oxygen steelmaking plants at Donawitz and Linz have proven successful. It is expected that this new steelmaking process will be utilized in areas where suitable raw materials are available as a supplement to conventional processes.

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Technical Note

G-Modulus Temperature Coefficient For Beryllium Copper Wire

by John T. Richards

ELASTIC constants, including E-modulus, G-modulus, Poisson's ratio, and compressibility, are affected by temperature, but not necessarily to the same degree nor in the same direction. Theoretical expressions relating the temperature coefficients for the various elastic constants have been introduced by Gruneisen.[1]

In considering the thermoelastic characteristics

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Table I. G-Modulus Temperature Coefficient

Source	Temperature Range, °F	G-modulus Coefficient, m*
Keulegan and Houseman[2]	−60 to +120	0.000334
Richards[3]	−95 to +32	0.000355
	+32 to +212	0.000389
Present study	−95 to +85	0.000256
	+85 to +212	0.000330
	+212 to +340	0.000407

* $m = \frac{1}{G}\frac{dG}{dT}$ where G is G-modulus, and $\frac{dG}{dT}$ is the change of G-modulus with temperature.

applicable in shear or torsion, the G-modulus* of most materials decreases with an increase in temperature. Since earlier tests[2,3] to determine the temperature coefficient for beryllium copper have been of rather limited scope and have not shown good agreement with theoretical considerations, additional work has been undertaken.

* Known variously as torsion, shear or Coulomb's modulus, or modulus of rigidity.

G-modulus values, in the present investigation, were calculated from the load-deflection characteristics of helical compression springs. Tests were limited to low stress ranges to minimize the effects of end coils and changes in pitch angle. Dead weight loading was applied, while deflection was measured with an electronic micrometer having a sensitivity of 0.00001 in. and an estimated accuracy of 0.000025 in. Springs were mounted in a suitable furnace permitting adequate temperature control over the desired test range.

Test material details were as follow: composition, 1.83 pct Be, 0.22 pct Co, 0.11 pct Fe, and the balance essentially Cu; diameter, 0.091 in.; cold drawn reduction, 44 pct; aging treatment, 2 hr at 600°F; and tensile strength, 206,500 psi.

The effect of temperature upon deflection rate (and also G-modulus) is apparent from Fig. 1. Plotted values represent the difference between observed values at a given load and those calculated from an assumed deflection rate. If actual and assumed deflections are identical, points fall on a vertical line. Variations are indicated by a sloping or curved line. In the case of a spring having greater stiffness (higher modulus) than the assumed deflection rate, points fall to the left of the vertical, while decreased stiffness or reduced modulus causes points to fall to the right. The average slope of the resulting curve is then applied as a correction factor to the deflection rate employed in the usual spring formula for determining G-modulus.

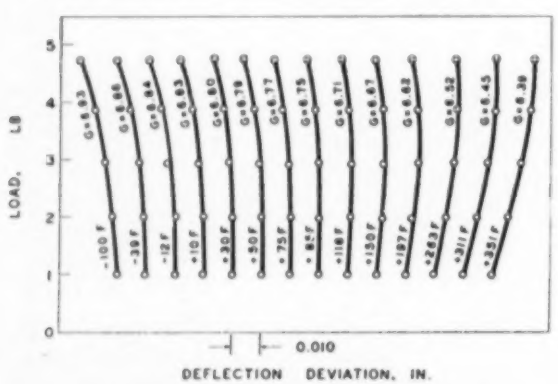

Fig 1—The effect of temperature upon the deflection characteristics of Be-Cu compression springs. The calculated G-modulus is shown.

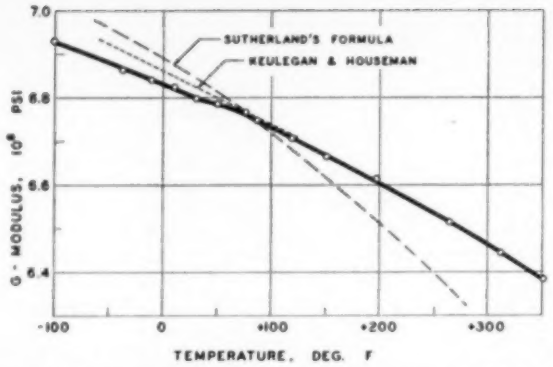

Fig. 2—The variation in G-modulus with temperature. Solid line is based upon data taken from Fig. 1.

Incidentally, this method of plotting is particularly effective in illustrating the practical significance of the temperature coefficient, since the change in stiffness of the spring with temperature is readily apparent. The curvature results from the decrease in the number of active coils resulting from the continuous closing of end coils with increasing load.

Although Fig. 1 shows the variation in G-modulus for rising temperature, tests were actually conducted over the complete temperature cycle. Good agreement was noted between rising and falling temperatures with no evidence of an open loop. Average temperature coefficients are listed in Table I.

A number of attempts have been made to relate the temperature coefficient with either the melting point or the thermal expansion coefficient of various metals and alloys. For example, Sutherland[4] has proposed the following formula:

$$\frac{G}{G_o} = 1 - \left(\frac{T}{T_m}\right)^2$$

where G is the G-modulus at any absolute temperature, G_o is the G-modulus at absolute zero, T is the absolute temperature at which G is to be computed, and T_m is the absolute temperature of melting point.

A curve corresponding to Sutherland's formula is shown in Fig. 2 drawn through the G-modulus value obtained from Fig. 1 at 75°F. The temperature coefficient according to Keulegan and Houseman[2] has also been superposed in Fig. 2 upon the curve representing the data from Fig. 1. The present data are considered more reliable than earlier values by the writer[3] because of the wider temperature range and greater number of experimental values. Furthermore, the increased deflection rates of the springs employed in the present tests mean improved sensitivity.

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[3] J. T. Richards: *Wire and Wire Products.* (1952) **27**, p. 304. Stamford. The Wire Association.

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On the Basic Bessemer Process

by Pierre Coheur and Hans Kosmider

New processes of blowing with an oxygen-enriched air or gas mixtures of oxygen and steam allow the steelmaker to produce, in a basic converter, a rimmed steel low in nitrogen (0.0020 pct), phosphorous (0.020 pct), and sulphur (0.020 pct). The physical properties (ductility) of the new steels are excellent. Furthermore, from the point of view of economy, these new methods of blowing offer outstanding possibilities.

THE natural occurrence of geological deposits of iron ore and local economic circumstances led European blast furnace men to work large quantities of lean iron ore with a phosphorus content as high as 0.5 pct. The iron obtained from these ores generally contains 1.5 to 2.1 pct phosphorus, 3.6 to 3.9 pct carbon, 0.3 to 0.6 pct silicon, 0.04 to 0.08 pct sulphur, and 0.3 to 1.2 pct manganese.

To produce steel from this pig iron, the steelmaker has at his disposal several processes. One of the simplest and most economical is the pneumatic process developed by S. G. Thomas in 1879. This process of steelmaking, called the Thomas process, consists of blowing through the bottom of a basic bessemer converter and is used widely in Western Europe, as shown in Table I. Due to the development of tonnage oxygen plants, this process has become more important because the use of oxygen offers new possibilities of steel manufacture. Not only may the quality of the steel be improved, but liquid pig iron also may be used, the composition of which does not meet the heat requirements of the blow with atmospheric air.

It is not intended to present here a detailed scientific discussion of the basic bessemer process but rather to discuss briefly its industrial possibilities in the case of rimmed steel and when blowing with atmospheric air, oxygen-enriched air, or gas mixtures of oxygen and steam. This paper is based on the current practice at Klöckner-Hüttenwerk Haspe's steel division, and at six Belgian and Luxemburg steel works. The latter are members of the Liege section of the C.N.R.M. (National Institute for Metallurgical Research) and are: ARBED, John Cockerill, Esperance-Longdoz, HADIR, Minière et Métallurgique de Rodange, and Ougrée-Marihaye, most of which are working with tonnage oxygen plants.*

In order to avoid any misunderstanding, it should be noted that most of the techniques to be mentioned, or similar ones, are used currently at many steel plants in Western Europe.

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Background

The basic converters used in Europe have a capacity of 17 to 65 tons of steel, and are similar in shape to the one in Fig. 1. The lining is brick or rammed dolomite and has a life of 250 to 500 blows. The bottom is rammed dolomite and has a life of 40 to 80 blows.

As far as we know, the type of blast used does not appear to have any influence on the life of the lining. Bottom life, on the contrary, depends very much upon the composition of the blast. It is only when

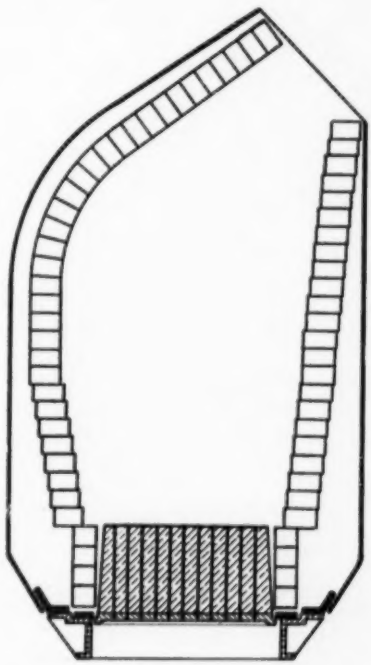


Fig. 1—Basic bessemer converter.

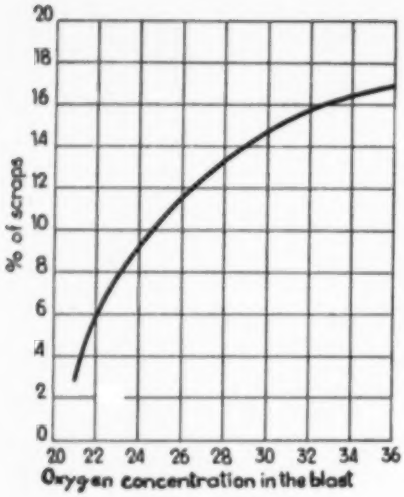


Fig. 2—Amount of scrap used to cool the bath when blowing with enriched air. Haspe.

resorting to special measures that the values indicated in Table II can be obtained. For example, in steel plant 3, bottoms had a mean life of 70 blows when blowing with atmospheric air. Since December 1952, this plant has been blowing all its iron with enriched air or with mixtures of oxygen and superheated steam. Results show that, when blowing with enriched air, the bottom life is 70 blows. When using both processes during the life of one bottom, 50 heats are blown with the superheated mixtures of oxygen and steam and at least 10 heats with enriched air.

In normal practice, liquid pig iron, lime for fluxing the silica, phosphoric acid, and scrap, ore, or limestone are employed. The amount of scrap, ore, and limestone depends on the oxygen concentration in the blast, the grade of steel to be produced, and finally on the economic conditions, Fig. 2 and Table III. For example, with atmospheric air the quantity of scrap used is 3 to 9 pct of the liquid iron. With enriched air, the quantity of heat developed during blowing is greater and this provides heat for melting additional scrap. With an oxygen concentration of 30 pct in the blast, the total amount of scrap charged may be increased to 15 or even 20 pct, i.e., approximately 0.3 lb of scrap for every additional cubic foot of oxygen added in the blast. When scrap is expensive, ore may be used. The equivalent factor, in respect to heat balance, between scrap and ore is approximately 2 to 2.8, which means that 1 lb of ore can be charged to the converter instead of 2 to 2.8 lb of scrap. A certain amount of limestone also may be charged instead of lime, and in this case, the equivalent factor is 1.8, which means that 1 lb of limestone can be charged instead of 1.8 lb of lime. With a mixture of oxygen and steam used from the beginning to end of the blow, the proper ratio of oxygen and steam may be chosen in order to melt the same amount of metallic additions melted with atmospheric air. In this case, the ratio of oxygen to

Table I. Steel Production in Western Europe, 1952*

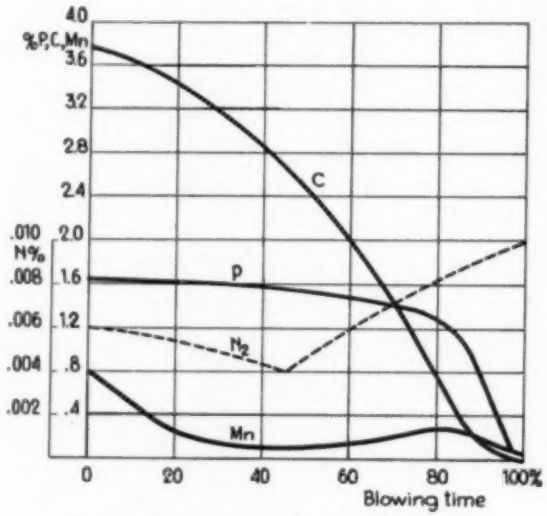
Country	Total Steel Production, Tons	Basic Bessemer Steel Production, Tons	Basic Bessemer Steel, Pct
Luxemburg	3,302	3,238	98
Belgium	5,608	4,631	82.6
Saar	3,105	2,319	74.7
France	11,934	7,265	61.2
West Germany	17,387	7,630	43.8

* In thousands of tons per annum.

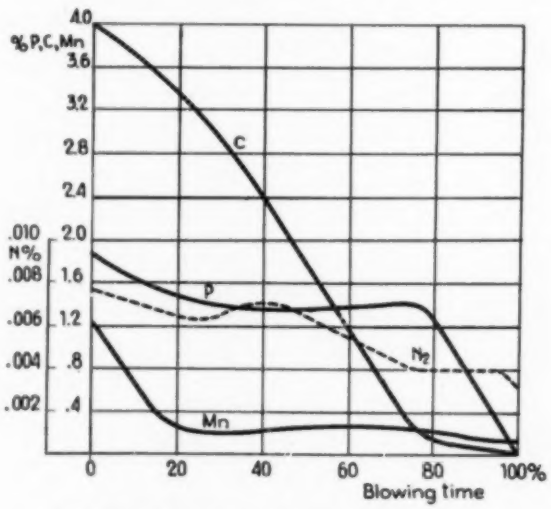
Table II. Lining and Life of Converter Bottom, Blow: Atmospheric Air

Steel Plant	Capacity of Converter, Tons	Lining	Life of Lining,* Blows	Thickness of Bottom, In.	Life of Bottom, Blows	Quantity of Dolomite, Lb per Ton**	Blowing Time, Min
1	22	Rammed	400 to 450†	25	60 to 70	18 per 22	15
2	30	Rammed	500 to 650†	36	55 to 65	18 per 22	21
3	15 to 16	Bricks	350 to 400	28	70 to 80	22 per 26	12
4	19 to 20	Bricks	250 to 300	24	55 to 65	18 per 22	15
5	16 to 17	Bricks	250 to 300	34	50 to 60	24 per 28	15
6	26 to 27	Bricks	300 to 500	36	75 to 85	16 per 20	16

* The life depends on the thickness of the lining.
** Pounds of dolomite used per ton of steel produced.
† With one repair.



a—Blow with an oxygen-enriched blast. Haspe.



b—Blow with a mixture of oxygen and steam. Haspe.

Fig. 3—Variation of carbon, phosphorus, nitrogen, and manganese content with the blowing time.

steam expressed in weight is approximately 1.7. It is, of course, possible to modify this ratio in order to decrease or increase the quantity of metallic additions to be transformed into steel.

In 1953, the steel plants mentioned blew more than 120,000 heats with enriched air or with the mixtures of oxygen and steam. The experience gained, and the economic factors involved, have supported the following four processes: 1—Blowing with enriched air and scrap. 2—Blowing with enriched air, scrap, and ore. 3—Blowing with enriched air, ore, and limestone. 4—Blowing with the oxygen and steam mixtures and scrap. With these four processes, the conversion time may be reduced by 30 to 50 pct of the time needed while blowing with atmospheric air. This provides a means of increasing production from 10 to 20 pct.

The iron yield (i.e., the quantity of steel produced in the ladle divided by the quantity of iron and metallic additions charged into the converter) is satisfactory for various types of blast and is 88.5 to 91 pct. The losses are due to the following: 1—The oxidation of the carbon, silicon, phosphorus, and manganese (a loss of 5.5 to 6.5 pct). 2—The iron in the slag which represents a loss of 2 to 2.4 pct.* 3—The ejections which cause a loss of about 1 pct.†

* 500 lb of slag per ton of liquid iron. Iron content of the slag is 10 to 11 pct.
† 50 lb per ton of liquid iron. Iron content of the ejections is 40 to 50 pct.

Quality of the Steel Produced

The quality of the steel produced with atmospheric air, or by one of the four processes mentioned, will be discussed from a chemical viewpoint.

Phosphorus: Whatever the type of the blast, the phosphorus concentration of the blown metal depends mainly (for the same slag basicity) on the bath temperature and its degree of oxidation. This degree of oxidation is generally estimated by taking into consideration the iron concentration of the slag as indicated in Fig. 4. In general, the bath temperature is about 2875° to 2975°F, and the blower stops the conversion when the phosphorus concentration is about 0.035 pct for which the iron content of the slag is around 10 pct, as shown in Fig. 5.

If reduction of the phosphorus is desired, it is only necessary to form a second slag. The first slag is the normal basic one. The blower stops the conversion when the phosphorus concentration is about 0.060 to 0.080 pct and pours off the slag. Afterwards, the blower charges the converter with a new basic slag (e.g., 200 lb of Na_2CO_3 for a 17 ton converter), turns the vessel upright, and blows for another 15 to 20 sec. The new slag is formed in a very short time, and the phosphorus content drops rapidly from 0.060 to 0.080 pct to 0.015 to 0.025 pct, Fig. 6a.

Table III. Raw Materials Used for Production of Converter Steel

	Atmospheric Air	Enriched Air, 30 Pct 1	2	3	Oxygen and Steam
Scrap, pct*	3 to 9	15 to 20	5.5 to 7.5	—	6 to 9
Ore, pct*	—	—	2 to 3	3 to 4	—
CaO, pct*	13 to 15	13 to 15	13 to 15	11 to 13	13 to 15
$CaCo_3$, pct*	—	—	—	3.5	—
Speed of conversion: tons per min for a 16 ton vessel	1.1	1.5	1.5	1.5	1.9
Oxygen consumption: cu ft per ton	—	730	730	730	1755

* Percentage of liquid pig iron.

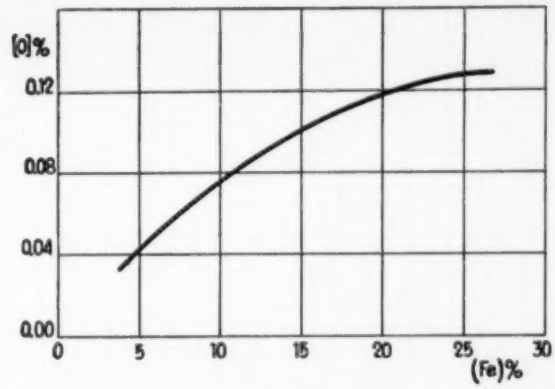


Fig. 4—Variation of oxygen in liquid steel with the iron content of the slag.

It seems that a correlation exists between the phosphorus content of the blown metal before and after the second slag. Indeed, it was found that ten points of variation in the phosphorus content before the second slag led to a variation of only one point after the second slag as shown in Fig. 6b. This standardizes the phosphorus content of the finished steel, Fig. 6a. It is worthwhile to note that the degree of dephosphorization obtained in a converter is approximately 99 pct.

Sulphur: The sulphur in the steel depends upon the sulphur content of the liquid pig iron and the degree of desulphurization during the conversion. In general the sulphur of the liquid pig iron can be regulated at the level desired by one or more treatments with soda ash in the iron ladle, as shown in Fig. 7. Another alternative is to mix the iron vigorously with the soda ash in the converter before charging the lime and the metallic additions. This reduces slightly the output of a converter but has the following benefits: 1—It reduces the cooling of the iron. 2—It increases the surface contacts between the slag and the iron. 3—It may be used in every steel plant, which is not the case for processes using the teapot ladle.

During conversion the degree of desulphurization depends primarily on the quality and quantity of the lime charged. In general this is 25 to 50 pct provided that the concentration of the sulphur in the lime does not exceed 0.150 pct and that the quantity of lime be sufficient to provide 10 to 15 pct free lime in the slag as shown in Fig. 8. The steel

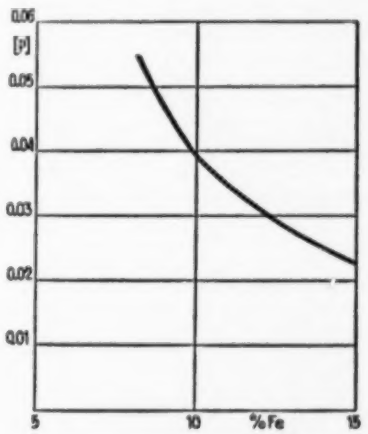


Fig. 5—Variation of phosphorus in steel with iron in the slag.

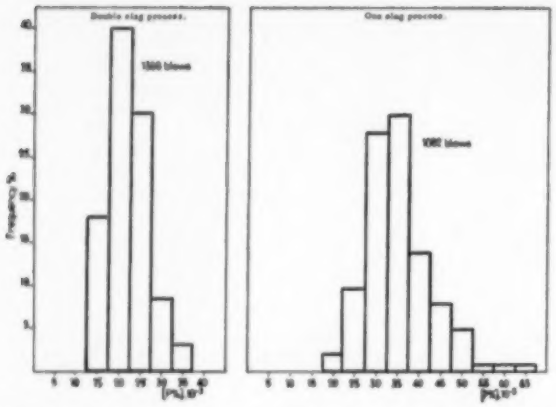


Fig. 6a—Frequency curve of phosphorus in the finished steel. Process with one or two slags. Esperance-Longdoz and John Cockerill.

produced has a sulphur content of 0.035 pct which, if necessary, can be reduced to 0.020 to 0.025 pct, as indicated in Figs. 9 and 10, or even below.

Nitrogen: Unlike phosphorus and sulphur, nitrogen in steel depends upon the blowing process. When blowing with atmospheric air, the nitrogen content varies between 0.008 and 0.014 pct. This rather broad dispersion is a consequence not only of variation in the quality of the iron but also of variation in temperature of the blown metal, mainly. For instance, in the case of bottom pouring of the steel, the temperature has to be 85° to 100°F higher than for top pouring, and the nitrogen is consequently 0.001 to 0.002 pct higher.

When the double slag process is used to reduce phosphorus, a higher temperature is needed to build the second slag. The nitrogen content of the steel is consequently higher, and this is one of the reasons why it is difficult, if not impossible, to consistently produce with atmospheric air a steel containing both low nitrogen and low phosphorus.

Fig. 11 is a frequency curve of the nitrogen content of one of C.N.R.M. steel plants (top pouring and one slag). The mean value of the nitrogen is 0.011 pct. To reduce this value, steelmakers have suggested several processes such as using ore, mill scale, or limestone instead of scrap. These processes are shown in Figs. 12 and 13.

Another way to reduce nitrogen, following Sieverts' law, is to lower the partial pressure of nitrogen in the blast. For this purpose the air is enriched with oxygen or, more drastically, the air is eliminated and the blast is blown with a mixture of oxygen and steam.

With the three processes using enriched air

Table IV. Steel Analysis: Mean Value, *M*, and Standard Deviation, *G*, of Steel Produced in Converter with Atmospheric Air, Enriched Air, and a Mixture of Oxygen and Steam

		Atmospheric Air	Enriched Air, 30 Pct Oxygen 1	2	3	Oxygen and Steam
Nitrogen	*M*	0.0110	0.0082	0.0063	0.0049	0.00225
	G	0.0025	0.0015	0.00125	0.0010	0.0004
Phosphorus, One slag	*M*	0.046	0.046	0.034	—	—
	G	0.0125	0.0123	0.0075	—	—
Phosphorus, Two slags	*M*	—	—	—	0.019	0.0195
	G	—	—	—	0.007	0.005
Sulphur	*M*	0.034	0.034	0.023	0.024	0.020
	G	0.0065	0.0065	0.0045	0.0055	0.003

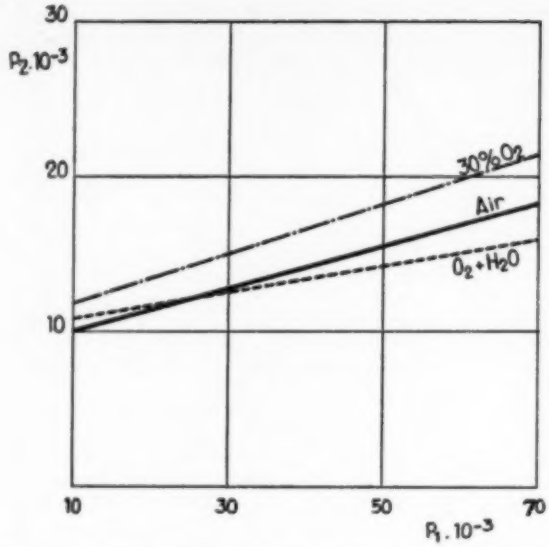


Fig. 6b—Relation between the phosphorus content before (P_1) and after (P_2) the second slag. Esperance-Longdoz and John Cockerill.

which were mentioned earlier, the steelmaker can reduce the nitrogen 20 to 60 pct and produce a steel with a nitrogen of 0.008, 0.006, or even 0.005 pct, which is comparable with open hearth steel, as shown in Figs. 11 and 14, and Table IV.

With the mixture of oxygen and steam, a steel is produced with a nitrogen content of 0.002 pct and a standard deviation as low as 0.0004 pct, which means that the production is more uniform and the ingot more homogenous, Fig. 11. With these mixtures of oxygen and steam, the influence of temperature disappears completely making it possible to blow a steel with both a low nitrogen and a low phosphorus concentration, which was considered impossible only three years ago. It is also interesting to note that the temperature of the blown metal may be rather high and, if necessary, brought between 3000° and 3100°F.

Carbon and Oxygen: It should be noted that the carbon in the blown metal is rather low and about 0.03 to 0.05 pct. If a higher carbon content is desired (i.e., 0.06 to 0.10 pct which is similar to the carbon content of open hearth steel), it is only necessary to add agents high in carbon, such as graphite, coke, etc., in the steel ladle. The oxygen content of any liquid steel depends on the carbon content, as is well known. It has been observed that whatever the type of blast used for blowing, the oxygen in the middle of the ingots is 0.015 to 0.025 pct for a carbon of 0.04 to 0.07 pct. This is similar to what is found in the case of the open hearth process. In other words, the relationship between the oxygen and the carbon, or the laws and constants of chemical equilibrium, are equivalent in both processes.

Manganese: Manganese in the liquid iron varies from 0.3 to 1.2 pct. During the first part of the blow, a certain amount is oxidized rapidly and enters the slag according to the laws of chemical equilibrium. The residual manganese in the blown metal may be 0.06 to 0.17 pct depending on the manganese content in the liquid iron, as shown in Fig. 15. Ferromanganese is added either in the converter or in the steel ladle in order to bring about deoxidation and to add the required amount. The manganese yield

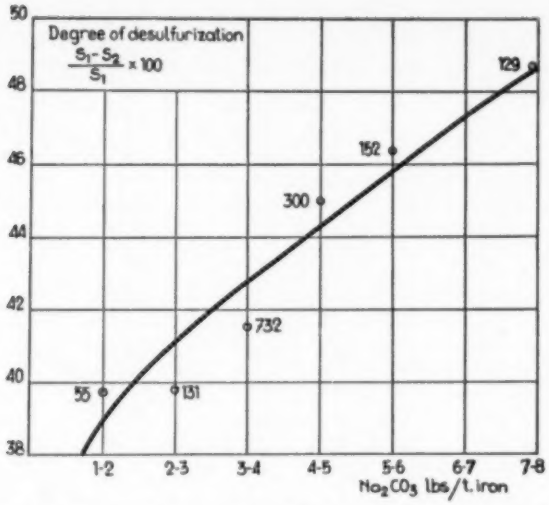


Fig. 7—Effect on degree of desulphurization of the amount of Na_2CO_3 used, two teapot ladles. The values near the dots refer to the number of charges treated. Arbed.

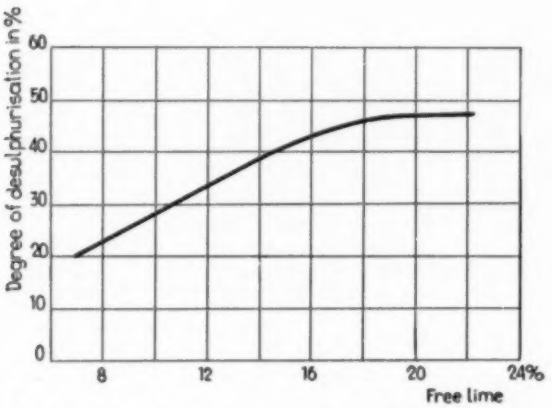


Fig. 8—Relation between the degree of desulphurization and the free lime content of the slag. Hadir.

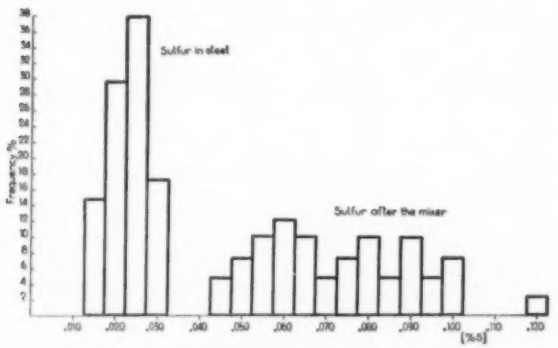


Fig. 9—Frequency curve of the sulphur content in the steel and the liquid pig iron. The pig iron is mixed with a soda slag in the converter. This pig iron has a manganese content of 0.80 to 0.90 pct. Ougree-Marihaye.

is 60 to 90 pct depending primarily upon the physical state of the ferromanganese added, the method of addition, and the chemical composition of the

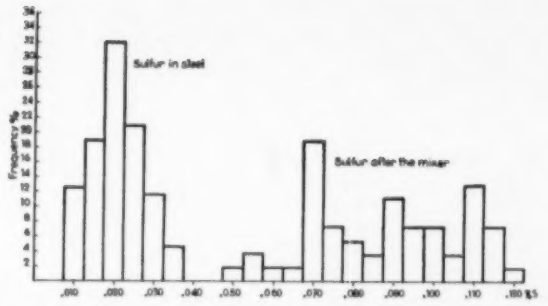


Fig. 10—Frequency curve of the sulphur content in the steel and liquid pig iron. The pig iron is mixed with a soda slag in the iron ladle, and its manganese content is approximately 0.80 to 0.90 pct. M. M. Rodange.

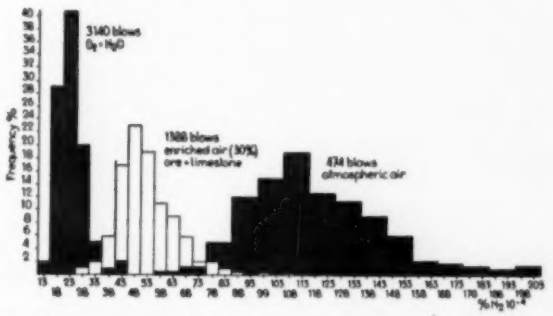


Fig. 11—Frequency curve of nitrogen in the steel when blowing with atmospheric air or a mixture of oxygen and steam. John Cockerill and Esperance-Longdoz.

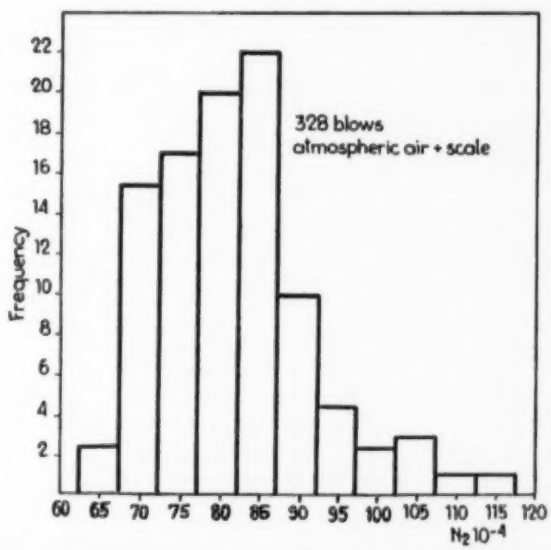


Fig. 12—Frequency curve of nitrogen in steel blown with atmospheric air. The bath is cooled with scale. John Cockerill.

blown metal. This loss is not necessarily dependent upon the nature of the blast used for blowing.

Physical Properties

The steel products obtained either with enriched air or with steam and oxygen mixtures have been subjected to physical tests and statistical studies

made at the steelworks or the Research Institute, and the results have been confirmed by consumers. Those tests, studies, and results are too numerous to be described here and will be published, in detail, in a separate paper. The only point that should be stressed is that improvement in ductility, which was the original and primary object of the use of oxygen, is obtained. Figs. 16 to 18 give indirect evidence of this. Steels produced by the methods described here have hardness and strength values, work hardening, cold bend, and drawability properties comparable to or better than values for normal open hearth steels.

Conclusion

From what has been presented, it appears that at the present time the steelmaker is in a position to choose the process of blowing to meet phosphorus, sulphur, and nitrogen specification requirements.

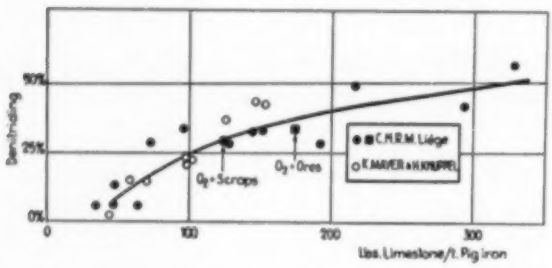


Fig. 13—Variation of denitriding of the blown metal with the amount of limestone charged in the converter. Ougree and John Cockerill.

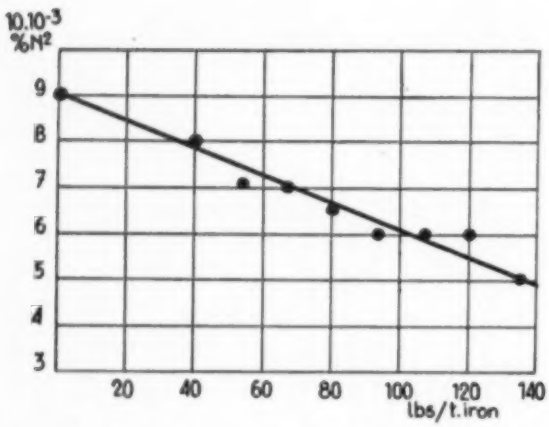


Fig. 14—Variation of nitrogen in the steel with the amount of ore charged in the converter. Ougree.

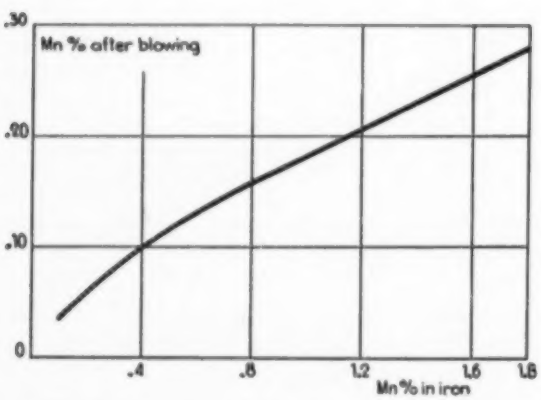


Fig. 15—Effect of manganese in liquid iron on residual manganese in the blown metal. Hadir.

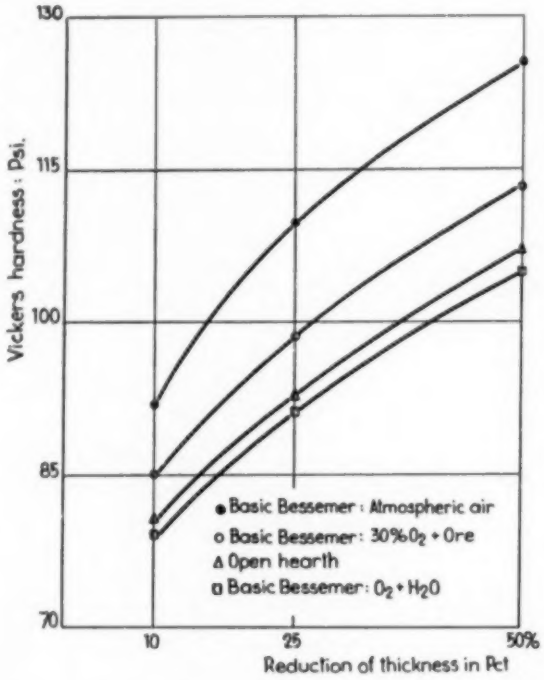


Fig. 16—Work hardening curve: Vickers hardness after aging vs reduction in thickness by compression.

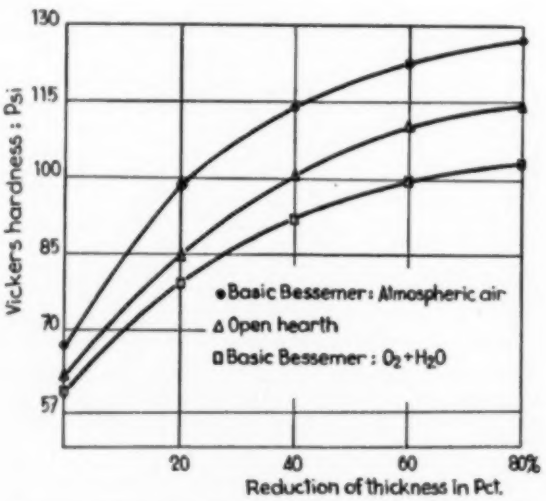


Fig. 17—Work hardening curve: Vickers hardness vs reduction in thickness by cold rolling.

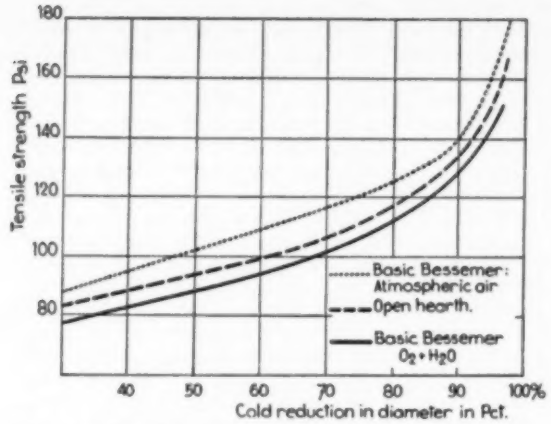


Fig. 18—Work hardening curve: tensile strength vs cold reduction in diameter by wire drawing.

One of the steel plants mentioned has made 3140 blows with oxygen and steam. The steel had the analysis given in Table V. This analysis, compared with blowing with atmospheric air, demonstrates the improvements in the pneumatic process. This also applies to the standard deviation which, in fact, is a measure of the production regularity. The ductility of these steels is excellent and at least equal to that of normal open hearth steel.

Table V. Analysis of Steel Blown with Oxygen and Steam

	Mean Value	Standard Deviation
Pct phosphorus	0.0195	0.005
Pct sulphur	0.020	0.003
Pct nitrogen	0.0023	0.0004

From the point of view of economy, these methods of blowing offer outstanding possibilities. Besides ensuring a steel of excellent quality, they increase the production capacity of converters while maintaining the principle of bottom blowing. Furthermore, they enable the use of pig irons of abnormal chemical composition which heretofore were either difficult to blow with the classic methods or which did not meet the heat requirements of the blow.

Forgeability of Steels with Varying Amounts Of Manganese and Sulphur

by C. Travis Anderson, V. V. Donaldson, Robert W. Kimball, and Francis R. Cattoir

THIS paper is the second of a series on the effect of various elements on hot-working characteristics of Fe-C alloys to be issued by this laboratory.[1] The investigations being conducted at the Ferroalloy Research Branch of the U.S. Bureau of Mines, Redding, Calif., involve fundamental research on the effects of various alloying materials and impurities on the physical properties of high purity Fe-C alloys. This paper deals with the determination of the limit of forgeability of steels containing varying amounts of manganese and sulphur. A paper discussing the hot-working characteristics and the physical properties of the forgeable alloys is being prepared.

One of the principal impurities in all steels is sulphur. The phenomenon, hot-shortness, commonly attributed to sulphur, has long been a source of expense and inconvenience to both the maker and user of steels. In steelmaking practice, this effect of sulphur has been overcome by the addition of manganese which, apparently, acts as a desulphurizer. Manganese is also used as a deoxidizer and alloying material. The critical and strategic position of manganese gives added importance to research which may contribute to its conservation and more effective utilization. It is not the purpose or intention in the present paper to give any explanation of the mechanism of either hot-shortness or its prevention but rather to report the results of forging high purity Fe-C alloys of the range of compositions near the limit of hot-forgeability. Temperatures for each phase of reduction were duplicated within 20° to 30°F for all tests.

The starting material was sponge iron, which, after melting and purification, gave a high purity iron containing about 0.001 pct each of silicon and phosphorus. Scrap iron was not considered suitable because of its high percentage of contaminants and wide variation in analyses. The sponge iron was melted and refined for melting stock in a three-phase, direct-arc, size ST, Moore Rapid Lectromelt furnace with a 500 kva rating as described in the preceding paper.

Additives were carefully selected to avoid addition of contaminants. The sulphur was added as resublimed flowers of sulphur or purified roll sulphur of analytical grade. The manganese used was dehydrogenated high purity electrolytic material produced by the Bureau of Mines plant at Boulder City, Nev. High purity aluminum in the form of wire was used for deoxidation. The limestone was obtained locally and had the following analysis: CaO, 53.7 pct; MgO, 0.8; ignition loss, 42.4; R_2O_3, 0.2; SiO_2 3.0; and P_2O_5, 0.014.

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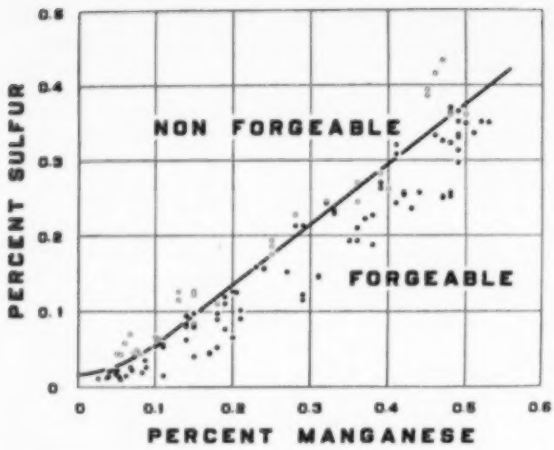

Fig. 1—Forgeable and non-forgeable alloys obtained by varying additions of sulphur and manganese to Fe-C alloys.

Since test results can vary considerably due to differences in pouring and fabricating procedures, this investigation was made on materials produced under carefully controlled conditions. Procedures were standardized insofar as possible to prevent variation in melting, pouring, forging, rolling, and testing operations.

For an average yield of 765 lb of metal, 850 lb of purified metal containing less than 1 pct C, 62 lb of limestone, 19 lb of lime (CaO), and 51 lb of raw ore were charged. The limestone was charged on the furnace bottom. During melting an effort was made to prevent the limestone boil until virtually all the metal had become molten. The oxidizing action of the strong limestone boil further lowered the carbon, silicon, and manganese content of the metal. After the limestone boil the slag was removed, carbon was added as coke or graphite, and lime was added to form a protective slag.

For steels containing up to about 0.25 to 0.30 pct S, additions of sulphur, aluminum, and manganese, in that order, could be added successfully to the ladle. With higher percentages of sulphur, recovery of the sulphur in the metal was erratic as a result of losses by evaporation before reaction with the melt. Therefore, it was found to be more practical to add the sulphur to the melt in the furnace just before

Table I. Compositions Near the Limit of Hot-Forgeability

Heat No.	Ingot No.	C, Pct	Mn, Pct	S, Pct	Hot Forgeable	Heat No.	Ingot No.	C, Pct	Mn, Pct	S, Pct	Hot Forgeable
147B	1	0.25	0.040	0.021	Yes*	202B	1	0.28	0.48	0.252	Yes
148B	1	0.25	0.045	0.022	Yes*	202B	2	0.28	0.48	0.257	Yes
149B	1	0.22	0.051	0.022	Yes*	203B	1	0.28	0.21	0.101	Yes
150B	1	0.19	0.049	0.017	Yes	203B	2	0.28	0.21	0.090	Yes
151B	1	0.29	0.062	0.015	Yes	204B	1	0.25	0.15	0.070	Yes
152B	1	0.22	0.053	0.014	Yes	204B	2	0.25	0.15	0.069	Yes
153B	1	0.19	0.055	0.010	Yes	205B	1	0.28	0.14	0.083	Yes
154B	1	0.25	0.067	0.070	No	205B	2	0.28	0.14	0.080	Yes
155B	1	0.26	0.060	0.058	No	206B	1	0.25	0.10	0.067	No
156B	1	0.21	0.076	0.049	No	206B	2	0.25	0.105	0.064	No
157B	1	0.28	0.087	0.035	Yes	207B	1	0.25	0.13	0.126	No
158B	1	0.19	0.068	0.026	Yes	207B	2	0.25	0.13	0.115	No
159B	1	0.27	0.070	0.022	Yes	208B	1	0.28	0.15	0.126	No
160B	1	0.22	0.17	0.044	Yes	208B	2	0.28	0.15	0.122	No
161B	1	0.22	0.083	0.19	Yes	209B	1	0.25	0.24	0.156	Yes
162B	1	0.27	0.15	0.040	Yes	209B	2	0.25	0.24	0.157	Yes
163B	1	0.24	0.11	0.015	Yes	210B	1	0.20	0.33	0.229	Yes
165B	1	0.20	0.040	0.014	Yes	210B	2	0.20	0.33	0.233	Yes
166B	1	0.21	0.026	0.011	Yes	211B	1	0.24	0.51	0.336	Yes
167B	1	0.20	0.037	0.012	Yes	211B	2	0.24	0.46	0.333	Yes
168B	1	0.20	0.088	0.026	Yes	212B	1	0.24	0.47	0.252	Yes
169B	1	0.18	0.14	0.062	Yes	212B	2	0.24	0.47	0.250	Yes
170B	1	0.19	0.20	0.065	Yes	213B	1	0.21	0.49	0.313	Yes
171B	1	0.21	0.15	0.084	No	213B	2	0.21	0.49	0.297	Yes
172B	1	0.22	0.19	0.076	Yes	214B	1	0.25	0.44	0.257	Yes
173B	1	0.25	0.18	0.052	Yes	214B	2	0.25	0.43	0.236	Yes
174B	1	0.23	0.18	0.110	No	215B	1	0.25	0.50	0.349	Yes
175B	1	0.20	0.17	0.045	Yes	215B	2	0.25	0.49	0.351	Yes
176B	1	0.19	0.23	0.159	Yes	216B	1	0.22	0.39	0.263	Yes
177B	1	0.25	0.25	0.194	No	216B	2	0.22	0.39	0.270	Yes
178B	1	0.26	0.25	0.186	No	217B	1	0.21	0.42	0.254	Yes
179B	1	0.24	0.28	0.228	No	217B	2	0.21	0.42	0.255	Yes
180B	1	0.20	0.25	0.176	No	218B	1	0.26	0.49	0.331	Yes
181B	1	0.22	0.36	0.310	Yes	218B	2	0.27	0.47	0.326	Yes
182B	1	0.25	0.38	0.188	Yes	219B	1	0.23	0.53	0.350	Yes
189B	1	0.23	0.105	0.057	Yes	219B	2	0.23	0.52	0.351	Yes
189B	2	0.23	0.10	0.054	Yes	220B	1	0.19	0.47	0.433	No
190B	1	0.21	0.15	0.097	Yes	220B	2	0.19	0.46	0.415	No
190B	2	0.21	0.14	0.095	Yes	221B	1	0.21	0.41	0.308	Yes
191B	1	0.22	0.18	0.089	Yes	221B	2	0.21	0.41	0.320	Yes
191B	2	0.22	0.18	0.097	Yes	222B	1	0.22	0.49	0.334	Yes
192B	1	0.19	0.19	0.119	Yes	222B	2	0.22	0.48	0.323	Yes
192B	2	0.19	0.19	0.110	Yes	223B	1	0.21	0.45	0.386	No
193B	1	0.21	0.11	0.056	Yes	223B	2	0.21	0.45	0.394	No
193B	2	0.21	0.11	0.053	Yes	224B	1	0.23	0.49	0.330	No
194B	1	0.22	0.08	0.044	No	224B	2	0.23	0.50	0.361	No
194B	2	0.22	0.08	0.044	No	225B	1	0.23	0.48	0.361	No
195B	1	0.21	0.05	0.044	No	225B	2	0.21	0.48	0.365	No
195B	2	0.21	0.055	0.043	No	226B	1	0.21	0.36	0.270	No
196B	1	0.20	0.20	0.126	Yes	226B	2	0.21	0.36	0.245	No
196B	2	0.20	0.205	0.125	Yes	227B	1	0.22	0.32	0.245	No
197B	1	0.24	0.29	0.213	Yes	227B	2	0.22	0.32	0.243	Yes
197B	2	0.24	0.28	0.213	Yes	228B	1	0.23	0.31	0.145	Yes
198B	1	0.25	0.36	0.193	Yes	228B	2	0.23	0.31	0.147	Yes
198B	2	0.25	0.35	0.193	Yes	229B	1	0.24	0.49	0.365	Yes
199B	1	0.22	0.42	0.257	Yes	229B	2	0.24	0.48	0.370	Yes
199B	2	0.22	0.41	0.243	Yes	230B	1	0.21	0.27	0.152	Yes
200B	1	0.21	0.38	0.227	Yes	230B	2	0.21	0.27	0.152	Yes
200B	2	0.21	0.37	0.222	Yes	231B	1	0.23	0.29	0.122	Yes
201B	1	0.15	0.40	0.262	Not	231B	2	0.23	0.29	0.115	Yes
201B	2	0.15	0.39	0.282	Not						

* Small ingots. Large ingots failed during forging tests before standard procedure was established.
† Analyses outside range considered.

the high sulphur heats were tapped. As little as a 5 gram addition of sulphur to the 850 lb charge could be detected by the analytical method used in the chemical laboratory. Aluminum just sufficient to prevent rimming action was added either to the furnace or ladle at the start of the tap. Manganese was added at intervals to the ladle in the metal stream during the early portion of the tap. Tapping was scheduled to obtain a final carbon content of 0.20 to 0.25 pct in most heats, the extreme analyses being 0.18 and 0.29 pct.

As a check on the possible effects of variation in aluminum content on test results, an additional weighed amount of aluminum wire was added to the mold of the second test ingot during pouring to increase the residual aluminum over that in the first ingot poured.

From each heat two test ingots of approximately 200 lb each, averaging about 7x7 in. cross section, were cast big-end-up with generous hot tops. Pouring usually was completed so as to leave a small skull in the ladle. After cropping the ingots a few inches below the shrink head, initial breakdown and forging to 2x2 in. bars were performed with an air-operated 1100 lb forging hammer. For forging and rolling, the ingots were heated to 2250°F in a butane-propane-gas-fired furnace and soaked at that temperature for at least 3 hr. In general, after light forging on all four sides and corners, each ingot was given an additional 30 min at 2250°F. Further reduction was performed with a minimum of reheats to maintain the temperature between 1800° to 2250°F, finishing at the lower temperature. Following the forging, the bars were cut to suitable length, reheated to 2250°F, and rolled to ¾ in. diameter rods, finishing temperatures being a maximum of 1600° to 1700°F. Form rolls having five diamond and five round passes were used for the rolling operation.

By controlled adjustment of the sulphur and manganese contents, a series of steels has been produced having compositions near the limit of forgeability. No known previous work of this nature and scale has been reported, and as a result much exploratory work was necessary before the trend of the curve representing the limit of forgeability could be definitely established. The excellent reproducibility and agreement of the results over the range of compositions studied appears to justify the work done in the preliminary phases of the investigations.

A series of controlled additions of aluminum was made to give ingot analyses ranging from 0.02 to 0.14 pct residual aluminum. Optimum addition of aluminum required for killing gave 0.06 to 0.08 pct residual aluminum. Although this amount may appear unusually high, it should be noted that aluminum was the only addition other than the carbon, sulphur, and manganese.

Segregation, nearly always found in steel ingots, made sampling of the ingots containing high percentages of sulphur quite difficult. No difficulty was experienced with regard to variation in carbon content and very little variation was found in manganese analyses of samples from different parts of an ingot. The uniformity of the manganese and carbon and also the lack of uniformity in the sulphur analysis in the two ingots from the same heat can be seen in Table I, which gives the chemical analysis of the ingots. Sulphur tended to exhibit greatest variation from the outside to the center of the cast ingot and, to a considerably lesser degree, from top to bottom of the ingot. Rods, however, after forging and rolling, gave satisfactorily uniform analyses.

As has been reported previously, high purity manganese-free steels containing over 0.017 pct S could not be forged successfully. Increasing the sulphur content by only 0.008 to 0.025 pct required the addition of 0.06 pct Mn before hot-forgeability was attained. This is illustrated graphically in Fig. 1. Alloys that broke up during forging or rolling and from which rod was not obtained are considered as not forgeable. Above 0.03 pct S, the manganese requirements for hot-forgeability follow a linear relationship which can be expressed by the formula:

$$\text{Pct Mn required} = 1.25\ (\text{pct S}) + 0.03.$$

This expression is valid under the test conditions up to above 0.375 pct S and 0.5 pct Mn, which were the limiting percentages used in the present work.

It may be noted that the term hot-forgeability as used in this paper refers only to the conditions of the laboratory test. It was possible to forge several compositions above the line determined by this work. However, variations of heating and forging procedures would have been required for each case, and this was not considered desirable for purposes of this investigation. The conditions of testing may be thought to be unnecessarily severe, but the remarkably consistent results obtained appear ample to justify adherence to the standardized program used.

Undoubtedly hot-shortness, which has been considered as due to a low Mn-S ratio should also be attributed to the effects of other minor constituents of the steel. Lack of fundamental knowledge regarding the exact role and effect of the individual elements on the properties of steel indicates the necessity for examining the effects many minor constituents exert on the forging characteristics of steel.

Summary

A series of high purity Fe-C alloys with carefully controlled additions of sulphur and manganese was prepared from sponge iron. A standardized procedure was established for melting, pouring, and fabrication by forging. Heats averaging 765 lb of metal product were produced in a basic-lined electric arc furnace. Two test ingots of about 200 lb each were poured from each heat. The compositions at the limit of hot-forgeability of Fe-C alloys containing manganese and sulphur in varying amounts has been determined. Manganese-free steels with over 0.017 pct S could not be forged. Above 0.03 pct S and 0.06 pct Mn, the relationship found can be expressed by the formula:

$$\text{Pct Mn required} = 1.25\ (\text{pct S}) + 0.03.$$

Variation in neither the carbon content within the limits tested nor the pure aluminum used for killing in this work had any significant effect on the limit of hot-forgeability as determined in this investigation. Every effort has been made to eliminate or control the effect of variables that could interfere in determining the value of this research.

Acknowledgments

Acknowledgment is made to R. G. Dutton, E. D. White, and R. E. Farnum who assisted in many phases of the preparation, melting, and pouring of the heats, as well as materially contributing to the handling, control, and fabrication of the test ingots.

[1] C. T. Anderson, R. W. Kimball and F. R. Cattoir: Effect of Various Elements on Hot-Working Characteristics and Physical Properties of Fe-C Alloys. *Trans.* AIME (1953) 197, pp. 525-529, JOURNAL OF METALS (April 1953).

Desilverizing of Lead Bullion

by T. R. A. Davey

IN 1947 the author became interested in the fundamental aspects of the desilverizing of lead by zinc, conducted some experimental work, and searched the technical literature for all available fundamental data. Since then a revival of interest in the subject in Europe resulted in the appearance of quite a number of papers. It became evident that it would be more profitable to collect together and examine thoroughly the results of various workers, than to attempt to duplicate the experimental determinations. There are many inconsistencies in the various publications, and it is opportune to review at this time the present status of knowledge on the Ag-Pb-Zn system. There is also a need for a clear description, in fundamental terms, of the various desilverizing procedures.

This paper is presented in four sections:

1—There is an historical review of the origins of the Parkes process, of the results of many attempts to find a satisfactory fundamental explanation for the phenomena, and of the modifications proposed to date.

2—A diagram of the Ag-Pb-Zn system is presented. This is believed to be free of obvious inconsistencies or theoretical impossibilities, although thermodynamic analysis subsequently may reveal errors.

3—The fundamental bases of the various desilverizing procedures, which have been used up to the present day, are described; and a new method is suggested for desilverizing a continuous flow of softened bullion in which the bullion is stirred at a low temperature in two stages producing desilverized lead at least as low in silver as that from the Williams continuous process and a crust which, on liquation, yields a very high-silver Ag-Zn alloy.

4—A suggestion is made for the revival of degolding practice, following a recently published account which does not seem to have attracted the attention it deserves.

The terms "eutectic trough" and "peritectic fold" as used in this paper are synonymous with "line of binary eutectic crystallization" and "line of binary peritectic crystallization" as used by Masing.[1]

The German literature on ternary and higher systems is rather extensive and a fairly general system of nomenclature has arisen, whereas in English usage the corresponding terms are not as well established. For this reason the meanings of terms used in this paper, together with the equivalent German terms, are given as follows:

1—Eutectic trough—eutektische rinne: line at which a liquid precipitates two solids S_1 and S_2 simultaneously. If the composition of a liquid which is cooling reaches this line, it then follows the course of this line until a eutectic point is reached, or until all the liquid is exhausted. The tangent to the eutectic trough cuts the line joining S_1S_2.

2—Peritectic fold—peritektische rinne: line at which a solid S_1 and a liquid L transform into another solid S_2. If the composition of a liquid which is precipitating S_1 reaches the line, on further cooling only S_2 is precipitated. The liquid composition moves from one phase region ($L + S_1$) into the other ($L + S_2$), and does not follow the course of the boundary. The tangent to the peritectic fold cuts the line S_1S_2 produced nearer S_2.

3—Liquid miscibility gap, or conjugate solution region—mischungslücke: the region within which two liquid phases coexist in equilibrium over a certain range of temperature. A system whose composition is represented by a point in this region comprises one liquid at high temperature; then as the temperature is progressively reduced, two liquids, one liquid and one solid, one liquid and two solids, and finally three solids.

4—Liquid miscibility gap boundary—begrenzung der flüssigen mischungslücke: the line along which the surface of the miscibility gap dome, considered as a solid model, intersects the surrounding liquidus surfaces.

5—Tie lines—konoden: lines joining points representing the compositions of two liquids, a liquid and a solid, or two solids, in equilibrium. In binary systems the only tie lines customarily drawn are those through invariant points, e.g., through the eutectics of the Pb-Zn and Ag-Pb systems, or the various peritectics of the Ag-Zn system, as in Figs. 1 to 3. In ternary systems it is desirable to draw sufficient tie lines to indicate the slopes of all possible tie lines.

6—Ternary eutectic point—ternäres eutektikum: point at which liquid transforms isothermally to three solids, S_1, S_2, and S_3. Such a point can lie only within the triangle $S_1S_2S_3$.

7—Invariant peritectic (transformation) point—nonvariante peritektische umsetzungspunkt: (a)—On the miscibility gap boundary, the point at which two liquids and two solids react isothermally so that $L_1 + S_1 \rightarrow L_2 + S_2$. (b)—On the eutectic trough, the point at which a liquid and three solids react isothermally so that $L + S_1 \rightarrow S_2 + S_3$. Such a point must lie on that side of the line joining S_2S_3 which is further from S_1. (c)—A further possibility, not found in this ternary system, is that the point is at the intersection of two peritectic folds when the reaction concerned is $L + S_1 + S_2 \rightarrow S_3$.

Historical Introduction

Karsten discovered in 1842 that silver and gold may be separated from lead by the addition of zinc.[2] Ten years later Parkes used this fact to develop the well known desilverizing process which bears his

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name.[3] A variation of the procedure prescribed by Parkes was patented by Balbach in the United States in 1864 and practiced by some American refineries for a few years.[4] However, the Parkes desilverizing process as generally practiced today differs little (except in engineering techniques) from the original: stirring-in of zinc, cooling and skimming crusts, liquation and then distillation of crusts, and cupellation of the rich retort bullion.

The correct theoretical interpretation of the phenomena occurring in the desilverizing process was not given for many years although various statements occur in the older text books, some partially correct, but many completely erroneous. The liquid miscibility gap which occurs in the binary Pb-Zn system, and hence also in the ternary system Ag-Pb-Zn, was usually considered to be bound up with the phenomena. The first determinations of its extent were made by Wright and Thompson in 1890.[5]

Kremann and Hofmeier[6] in 1911 were the first to show that the liquid miscibility gap has no bearing on the practical process, and later Hofman[7] made this fact generally available to members of the German-reading world. The liquid miscibility gap of Bogitch[8] in 1915, was much less accurately determined.

Williams in 1920,[9,10] independently of Kremann and Hofmeier whose work was unknown to him, showed that in desilverizing processes the liquid miscibility gap is not entered by the bullion composition, but that solid Ag-Zn phases are rejected by the cooling bullion or "mother liquor." He redetermined the miscibility gap and determined for the first time the progressive changes in composition of lead containing zinc and silver as the temperature is lowered. These changes may be followed on his cooling curves.

Williams also determined, although no claim of great accuracy was made, the liquidus isotherms in the region of the lead corner and the course of the eutectic trough which runs between the eutectic points on each of the systems Pb-Ag and Pb-Zn.

The practical applications by Williams of his theoretical findings should be too well known[11-30] to warrant more than brief mention here. He used his cooling curves first for the accurate determination of the most economical zinc additions to be used for the two-stage batch desilverizing process; his liquid miscibility gap and tie lines for the development of

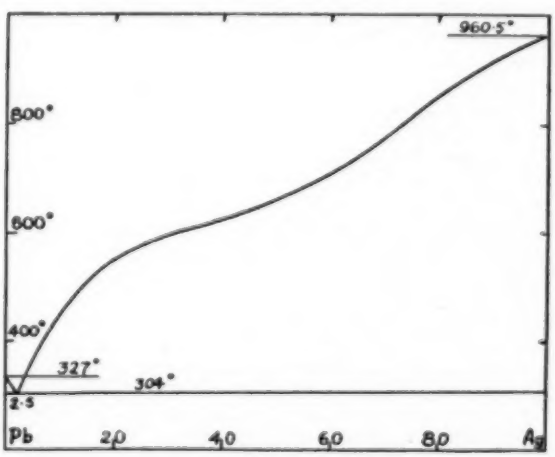

Fig. 1—Binary diagram of Ag-Pb.[22]

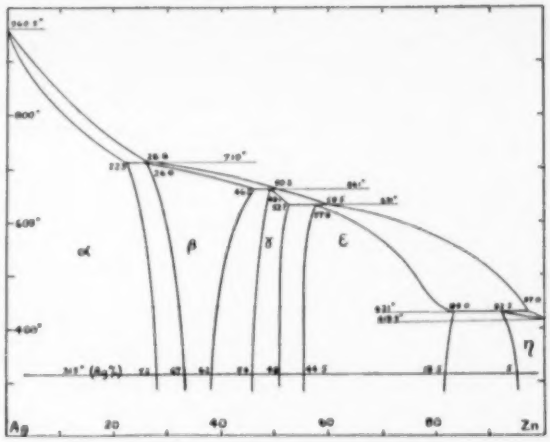

Fig. 2—Binary diagram of Ag-Zn.[22]

the crust enrichment process; and finally, his knowledge of the whole system for his now famous one-stage continuous desilverizing process which he termed the "conjugate solution process."

It is of interest to note that Williams was not the first to enrich Parkes crusts by liquation, although he was apparently the first to liquefy the zinc-rich phase as well as the lead-rich phase. Lead was liquated from Parkes crust in Europe[2] and the United States[4] before the advent of the Howard press. After the introduction of this press, the liquation of crusts was almost universally discontinued, but still persists in a few cases.

In 1947 Jollivet[31] made a more accurate determination of the eutectic trough of the ternary system. His interpretation of the results was based on the assumption that the intermediate phases of the Ag-Zn system are of fixed compositions, whereas in fact each exists over an appreciable range of composition.

In 1948 Henglein and Koester[32] examined the entire ternary system again determining, in particular, the critical and invariant points on the liquid miscibility gap. Their diagram appears to be the first in which all the phase boundaries in the system are represented correctly. Only two minor details of their results cannot be reconciled with the other data collected here: their Fig. 20 depicts the eutectic trough as sloping continuously downward from the Pb-Zn eutectic to the Pb-Ag eutectic, and the ϵ-η invariant point on the miscibility gap is too far from the Pb-Zn side to be consistent with all other work dealing with the lead-rich corner.

In 1950 Johannsen and Lange-Eicholz[16] re-examined equilibrium relations in the field where Williams' cooling curves were determined, that is, between the liquid miscibility gap boundary and the eutectic trough in the lead corner. Their experimental method was most ingenious: holding lead at constant temperature in an Ag-Zn crucible of known composition and sampling the lead solution after it was presumed equilibrium had been established. This method should give very accurate determinations of the liquidus isotherms but, unless the tie lines that join the crucible (Ag-Zn solid phase) compositions and the lead saturated solution compositions pass through the lead corner when produced, it is doubtful whether true equilibrium could result in a reasonable time. In this system only one such tie line produced does pass through the lead

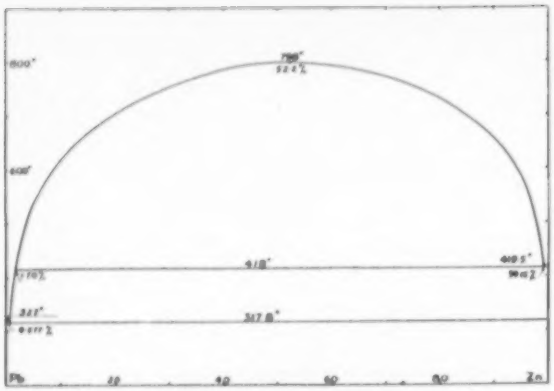

Fig. 3—Binary diagram of Pb-Zn.[22]

corner.* In all other cases the walls of the crucible have to change in composition to achieve equilibrium with the lead solution, and only after diffusion in the solid walls has re-established their homogeneity, could true equilibrium be reached. Then the composition of the crucible should be redetermined for the results to be accurate. The implicit contradiction between these laboratory tests and results of some carefully controlled plant tests is ignored by these authors although it is evident if their Figs. 1 and 6 are drawn on the same co-ordinates.

* One tie line at each temperature. The tie line to the maximum on the eutectic trough must pass through the lead corner when produced.

The position of the eutectic trough found by Johannsen and Lange-Eicholz is stated to confirm Jollivet's results although their drawings show it in a rather different position. Finally, Johannsen and Lange-Eicholz found a ternary eutectic point near the binary Pb-Zn eutectic which, however, is geometrically impossible, since this point lies outside the triangle joining the limits of composition of the $\epsilon + \eta$ region with the lead corner. Assuming that their "line of doubly saturated solution" does in fact represent a phase boundary, then it is not a eutectic trough, but a peritectic fold where the reaction is $L + \epsilon \rightarrow \eta$. The boundary meets the eutectic trough in an invariant peritectic transformation point where $L + \epsilon \rightarrow \eta +$ Pb, and not in a ternary eutectic point.

In 1951 and 1952 further determinations within the miscibility gap were reported.[17,18] These are in a region which is not of present practical importance and will not be discussed here. The results may well provide valuable material for thermodynamic calculations on the system.

A most significant paper[19] appeared in 1952, following the application for a German patent in 1951,[20] describing a new method for batch desilverizing of lead by a low temperature, practically isothermal, process. The zinc addition is made in two stages, but the process is conducted in such a way that the time requirement could not be reduced by making the process one stage.

A similar method had already been used in laboratory tests by the present author, with the object of providing data which could be used to propose a continuous low temperature desilverizing operation more suitable than the Williams process for treating small lead production rates† by virtue of lower maintenance and fuel costs. The intention of the author clearly has been anticipated by Lange-Eicholz and Wasow[19] although it is not suggested in their paper that they realize that their process, with slight modifications, could be operated on a continuous flow of bullion.

† The Williams process has been used commercially at Port Pirie, Australia for lead flows from 10 to 37 tons per hr, which even near the lower limit are above the capacities of many plants. With the smaller flows the practical operating difficulties are greater, and maintenance costs per ton of bullion are higher.

The Ag-Pb-Zn Phase Diagram

The binary diagrams Ag-Pb, Ag-Zn, and Pb-Zn are shown in Figs. 1 to 3. Critical or invariant temperatures and compositions are marked on them.

The representation of any ternary diagram as complicated as the Ag-Pb-Zn system is difficult. If all solidus and liquidus boundaries are shown, the diagram becomes almost unintelligible. It is perhaps due mainly to the difficulty experienced in understanding other authors' representations and descriptions that so many publications relating to this system reveal apparent neglect or misinterpretation of what has gone before.

The present paper attempts to deal with the difficulty in the following way: Scale drawings are presented, showing the entire diagram on an equilateral base with only liquidus phase boundaries and isotherms marked; the lead side and the zinc side of the liquid miscibility gap on rectangular co-ordinates; and two greatly magnified sections of the lead corner on rectangular co-ordinates. Then schematic representations, not to scale, are used to illustrate the course of a number of desilverizing procedures.

Fig. 4 shows the projection of the liquidus phase boundaries and isotherms on to the base of the triangular prism, which is required to represent the solid model of the system if temperature be included as the third (vertical) dimension.

The central liquid miscibility gap, or conjugate solution region, is crossed by a number of tie lines to show the compositions of solutions mutually in equilibrium. The tie lines drawn are those connecting the peritectic invariant points on either side of the miscibility gap. The invariant points are shown with greater clarity in Figs. 5 and 6, which respectively show the lead and zinc corners, plotted rectangularly.

The border system temperatures and compositions taken are those from Figs. 1 to 3. The border of the miscibility gap is taken from Kremann and Hofmeier,[9] as are the tie lines, with one exception. The lowest tie line at 425°C is drawn to the lead-rich boundary at the point found by Johannsen and Lange-Eicholz[20]

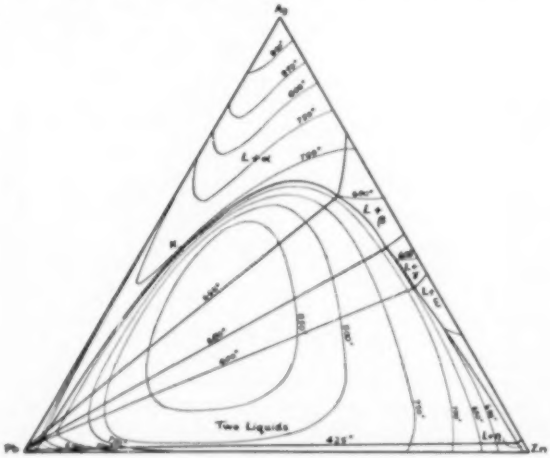

Fig. 4—Ternary diagram Ag-Pb-Zn.

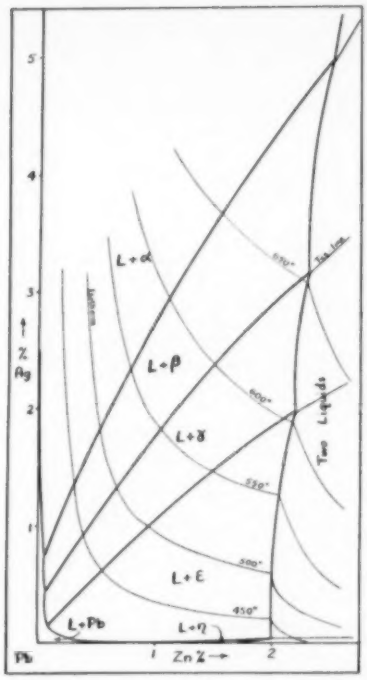

Fig. 5—Pb-Ag side.

for the intersection of the miscibility gap and their line of doubly saturated solution. The temperature which Kremann and Hofmeier quote for this point, namely 425°C, fits in quite well with Johannsen and Lange-Eicholz's diagram of the lead corner. The isotherms drawn inside the miscibility gap are taken uncritically from the paper by Seith and Helmold.[17]

The lead corner is shown in two different magnifications in Figs. 7 and 8, to scale, and schematically in Fig. 9. It is not possible here to describe fully the processes by which the invariant points were reached. In brief, the isotherms and eutectic trough of Johannsen and Lange-Eicholz[18] were used. The data of Jollivet,[14] together with the cooling curves of Williams[5] which are confirmed by Johannsen and Lange-Eicholz's plant tests,[18] have been used to determine the positions of invariant points on the eutectic trough. All data have been accepted unless contradicted by apparently more accurate work or unless internally inconsistent. No definite degree of accuracy can be claimed therefore for the diagram, but it is the best that the author could construct, taking into account all available published work, and avoiding theoretical impossibilities as they were recognized.

Some further mention should be made perhaps of the peritectic folds. The $L + \epsilon$, $L + \eta$ boundary is a true peritectic fold at all points, where $L + \epsilon \rightarrow \eta$. The other boundaries ($\alpha$, β, γ, ϵ) are actually eutectics for an undetermined extent in the centers. At their upper extremities the peritectic reactions are of the type $L + \alpha \rightarrow \beta$, and the temperatures fall toward the Pb-Zn side. At their lower extremities the reactions are of the type $L + \beta \rightarrow \alpha$, and the temperatures fall away from the Pb-Zn side.

No temperatures have been shown on the eutectic trough, since the only information available is that it is practically an isotherm at 315°.[14] It is theoretically necessary, however, that the trough has a temperature maximum in the region sloping downward both past the $\epsilon - \eta$ peritectic transformation point to the Pb-Zn eutectic, and past the $\epsilon - \gamma$, $\gamma - \beta$, and $\beta - \alpha$ peritectic transformation points to the Ag-Pb eutectic. This fact has not been recognized in previous publications of this diagram, but arises from consideration of a few geometrical necessities of the graphic representation of phase equilibria, in conjunction with the cooling curve data.[9,14,18]

A further point of minor interest is that, on cooling a lead bullion containing zinc and silver, the composition changes along a curve concave upward as Williams has shown, not in a straight line as almost all other writers have assumed. The theoretical necessity for this can be appreciated by considering the binary Ag-Zn diagram. While any solid phase is crystallizing from a cooling liquid, the solid phase is richer in zinc and poorer in silver, the lower the temperature. In the ternary diagram, the tangent to a cooling curve must point to the composition of the Ag-Zn alloy being precipitated,‡ and since the alloy is poorer in silver the lower the temperature, the slope of the tangent must be less at lower temperatures; and thus the cooling curve is concave upward.

‡ This is strictly true only for nonequilibrium cooling, in which a solid phase once precipitated does not react continuously with the remaining liquid, and progressively change its composition. (Nonequilibrium cooling occurs in practice.) But even if solid phases already precipitated were to react with the liquid so that the final solid product would be homogeneous and in equilibrium with the final liquid to solidify, then the cooling curve would still not be a straight line.

It should be observed that whenever a cooling curve cuts the boundary between ϵ and η phase precipitation regions, there will be an abrupt change of slope. That means that the crust formed just above the temperature of the intersection will be much higher in Ag-Zn ratio than that just below.

There is practical significance in the fact that this boundary intersects the eutectic trough in a peri-

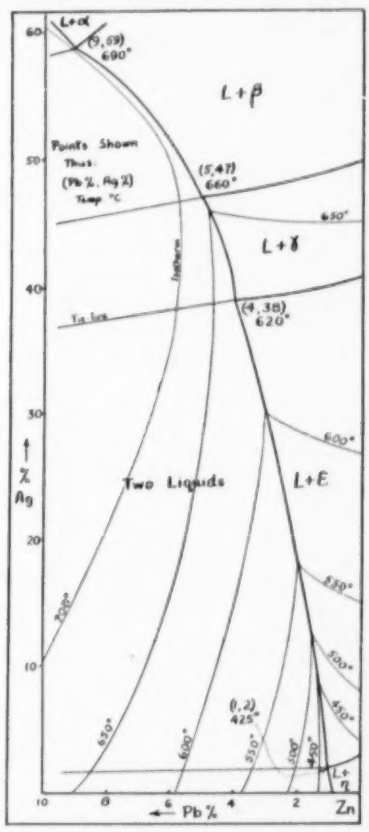

Fig. 6—Zn-Ag side.

tectic transformation point, and not in a ternary eutectic point. If the point were a eutectic, its composition would be the lowest silver concentration obtainable in lead by making zinc additions in any manner. Lange-Eicholz and Wasow[29] erroneously consider that their process would overcome this fundamental difficulty. There is in fact no limit to the degree of removal of silver from lead. By repeated zinc additions, silver may be reduced to any desired figure, since the Pb-Zn eutectic is approached with every treatment. At Port Pirie assay lead is desilverized to less than 0.000032 pct Ag by successive batch treatments.

In conclusion, although the fact may not be of practical significance, it is theoretically possible to dezinc lead by addition of silver, and zinc may be reduced to any desired figure by repeated treatments, approaching the Ag-Pb eutectic in this case.

Practical Desilverizing Processes

It is possible to desilverize lead to very low limits using any number of stages, from one up, of zinc addition. The greater the number of stages, the less is the total zinc consumption in production of Ag-Zn alloy or crust (provided that crusts are returned from each stage to the preceding one) and the lower the temperature required for stirring in each stage. But the time requirement and the zinc loss by oxidation are both greater. The zinc consumption for saturation of the desilverized lead is, of course, independent of the number of stages.

The practical difficulty which precludes the use of only one stage in most cases is that of dissolving all the zinc required at a high temperature without severe loss by atmospheric oxidation.[25] For example, a lead containing 0.2 pct Ag (65 oz per long ton) requires about 2 pct Zn to be stirred in at a temperature exceeding 450°C, in order that the lead when cooled to its freezing point shall contain about 0.0005 pct Ag (0.16 oz per long ton).

The two-stage process, in one form or other, is almost universally used. A third stage of treatment is conducted whenever the silver content is still too high after the second stage.

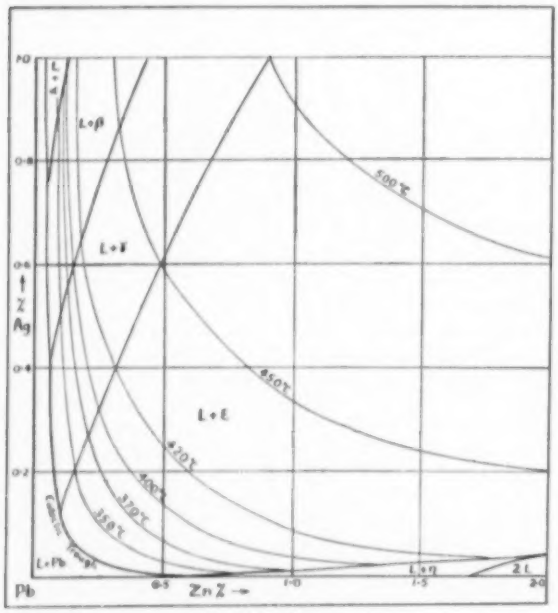

Fig. 7—Lead corner.

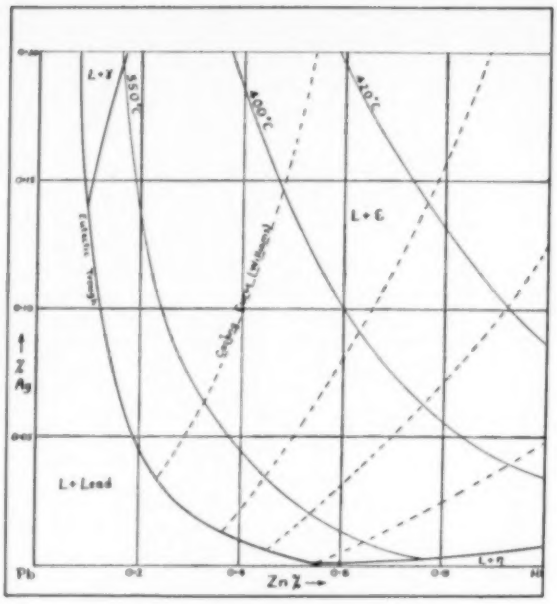

Fig. 8—Lead corner, further magnified.

Batch Processes, Using Zinc in Two Stages: The possible varieties of this procedure have been discussed so thoroughly by Williams[9] that it is unnecessary to repeat here at length the advantages and disadvantages of each variation. The three main ones are illustrated schematically in Fig. 10, and for convenience are designated "Pirie", "American," or "German." The Pirie method has been reported frequently,[25-27] the German and American methods have been discussed.[16]

Each of the two stages comprises two steps: the first step being stirring in of zinc, the second step cooling of the bullion and skimming the crust. In all three methods, in the first stage, second crusts are stirred into softened bullion (step 1), the bullion is cooled and skimmed (step 2), and in the second stage new zinc is stirred in (step 3), and bullion is cooled to its freezing point and skimmed (step 4).

In the Pirie and American methods, no new zinc is added in the first stage (step 1), whereas a small amount is added in step 1 of the German practice.

In the Pirie and German methods, the cooling in step 2 is carried nearly to freezing point, while in the American, cooling is stopped at about 420°C.

The best practice is seen to be the return of second-stage crusts to the bullion in the first stage, stirred in at a temperature sufficient to hold all the zinc and silver in solution, and cooling of bullion in the first stage to the lowest possible temperature consistent with the removal of the rich crust relatively free from entangled lead. Enough zinc is added in the second stage (at a temperature sufficient to dissolve it completely) to reach a cooling curve which intersects the eutectic trough at a silver value below the standard set for market lead.

If the bullion entering the desilverizing stage is relatively constant in silver composition from batch to batch, the zinc addition in the second stage will be $(Z + A/R)$ pct of the bullion weight, where Z is the percentage of zinc content remaining in the desilverized lead, A is the percentage of silver in the initial bullion to be desilverized, and R is the de-

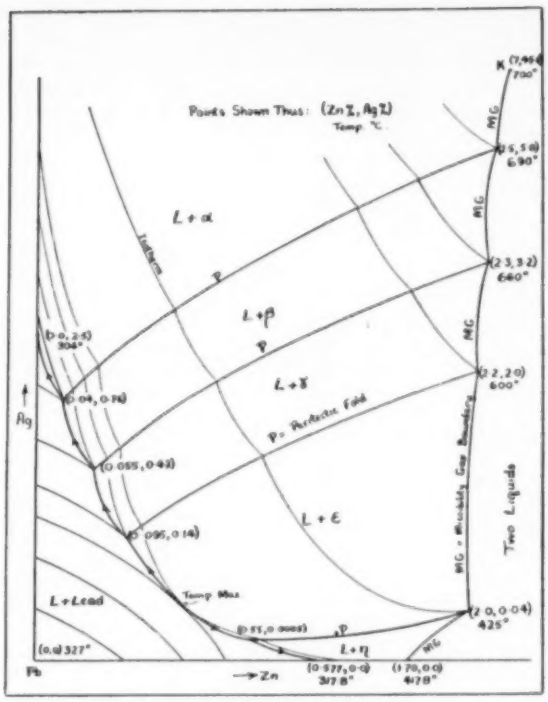

Fig. 9—Lead corner, schematic.

sired ratio of silver to zinc in the rich crusts produced in the first stage.

If the bullion varies in silver content from batch to batch, the zinc additions required for a number of procedures are listed in the very complete zincing tables of Williams.[9] These tables appear to be the only ones published although many plants have compiled their own for the particular variation practiced.

In the first stage an ϵ-phase of high Ag/Zn ratio is precipitated from the bullion while cooling, and in the second stage an ϵ-phase of low Ag/Zn ratio. It is not necessary to operate so that the still lower grade η-phase is precipitated. A second (poor) crust of Ag/Zn ratio of about 18.5/81.5 = 0.227 may be obtained while leaving only 0.0005 pct Ag and 0.55 pct Zn in the desilverized lead if care is taken to have the lead just at its freezing point while the final skim is made. In practice it is, of course, preferable to have a little more than this quantity of zinc remaining in the desilverized lead, so that the finishing temperature is not so critical.

If less than 0.0005 pct Ag in desilverized lead is desired, then it is necessary that the cooling curve enters the phase region $L + \eta$. At least some of the final crust will have an Ag/Zn ratio of less than 5/95 = 0.0526, so that the rich crust produced will also have a lower Ag/Zn ratio than the previous case, and the overall zinc consumption will be greater.

For a given silver input analysis, increasing the zinc addition to the second stage lowers the attainable final silver analysis. Economic considerations determine the point at which it would be preferable to use three zincings, rather than increased zinc addition, to obtain a lower final silver analysis.

Fig. 11 illustrates the differences between desilverizing operations with one, two, and three zincings, designated I, II, and III, respectively. The subscript m_1 indicates first mixing step, c_1 first cooling step, etc. It is assumed that third-stage crust is returned without new zinc addition to the second stage, and second-stage crust without new zinc addition to the first stage.

Since the total zinc consumption, as previously stated, is Z + A/R and both Z and A are independent of the number of zincings, the only variation that can occur is in R, the ratio of silver to zinc in the first-stage crust. Fig. 11 shows how this ratio is higher the greater the number of stages there are, since the cooling curve IIIc$_1$ is steeper, and Ic flatter, than IIc$_1$.

The recirculation of crusts, in effect, concentrates the silver from several batches of lead into the one batch, so that a higher Ag/Zn ratio is obtained in the lead, and a higher Ag/Zn ratio results in the crusts formed.

A many-stage recirculation procedure to build up a silver content of hundreds of ounces per ton was patented some years ago in this country,[24] but is apparently no longer practiced.

Isothermal Low Temperature Batch Process: This process is fully described in the German literature[10, 20] but will be described here briefly because, to the author's knowledge, no English reference has yet been published.

Fig. 12a shows, on the 330°C isothermal phase diagram, the course followed by bullion composition during the process. (The fact that the bullion temperature rises from 330°C to 340°C and then falls to 320°C can be ignored for the sake of simplicity.) The original bullion is placed in a kettle in which the second crust of the previous charge remains. One third of the total zinc addition is made while the kettle is stirred, the bullion temperature being about 330°C. The crust which results after this stirring is removed for liquation, after which it contains about 30 pct Ag and 60 pct Zn. The remaining two-thirds of the zinc is then added, with continued stirring, and the bullion finally is allowed to cool from 340° to 320°C, just above its freezing point. When the fine crystals of the Ag-Zn alloy have floated out, the desilverized lead is tapped from the bottom of the kettle, leaving the second crust as a solid bridge on the kettle.

The process from start to finish requires only 2 to 2½ hr, and the silver kettle itself requires no fuel as hot bullion from softening is used to melt the

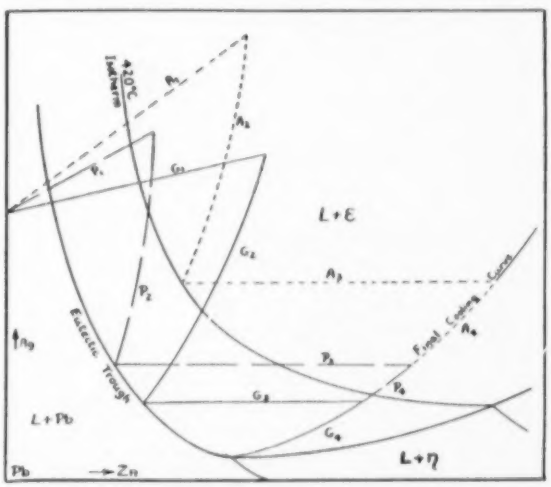

Fig. 10—Comparison of two-stage batch processes.

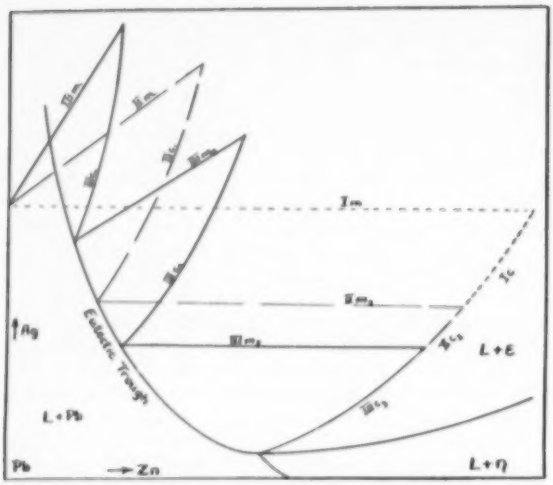

Fig. 11—Comparison of one, two, and three-stage batch processes.

second crust from the previous charge. Zinc is added as a saturated solution in lead at 550°C, the quantity of lead required for this purpose being one-third to one-half the amount of that to be desilverized. In order to prevent the loss of 0.54 to 0.58 pct Zn in this extra lead, a portion of the desilverized lead is pumped back to the solution kettle to dissolve the zinc required for the next charge.

The qualitative illustration of what occurs in each stage of the process is given in Fig. 12b, which again shows the isothermal section at 330°C of the phase diagram, but for the sake of intelligibility omits showing all the tie lines drawn in Fig 12a. The second zincing will be illustrated first.

The zinc solution in lead, S, is added to bullion B_1, from which the first crust has been removed. The resulting mixture is M_2. At equilibrium this mixture consists of B_2 and A_2, the liquid bullion and the solid Ag-Zn alloy, respectively. The stirring ensures that something approaching equilibrium is established. B_2, the desilverized lead, is tapped out of the kettle, leaving behind the alloy A_2 with some entangled B_2, the resultant crust having the composition C_2; this is the second crust.

Now the initial bullion B is added to the crust C_2, the resultant mixture being M'_1, and the portion of zinc in lead solution, S, for the first zincing is added, yielding mixture M_1. After stirring to produce equilibrium, the products are bullion B_1 and alloy A_1. The high grade or first crust, C_1, consists of alloy A_1 and entangled bullion B_1, which may be partially removed by liquation.

Two criticisms may be leveled at the process as described:[19] 1—There seems to be no advantage in adding new zinc to the first stage. The only advantage in the old batch method is that a reduced quantity of second crust must be skimmed off and returned to the first zincing kettle. Here no skimming of the second crust is required, so it should be of advantage to add all the new zinc in the second stage, thereby achieving either lower silver values in the desilverized lead, or else a higher Ag/Zn ratio in the first crust skimmed, and therefore lower zinc consumption. 2—There seems to be no overall advantage in using a solution of zinc in lead rather than solid zinc additions. Rather more vigorous stirring would be required to react solid zinc than a Zn-Pb solution, but as against greater stirring charges, the absence of a solution kettle and also nonretention of one-third to one-half of the desilverized lead as a solvent for zinc for the next charge must be credited.

Before reading ref. 19, the present author had, in laboratory tests, desilverized 5000 gram charges of bullion from 60 oz per long ton silver to a fraction of an ounce per ton, by stirring in solid zinc at 350°C for 1 hr, and cooling to the lead freezing point.

Williams Crust Enrichment Process: In 1923 Williams developed his crust enrichment process for working up the rich Parkes crusts mainly by separating entangled lead. His process differed from liquation processes which had been used many years previously in that, instead of merely raising the temperature of the crust sufficiently to melt out the lead and leave the Ag-Zn alloy solid, Williams melted both phases and thus obtained conjugate solutions. Thus a more complete separation of the two phases (lead-rich and zinc-rich) is possible by using higher temperatures than previously.

It was found that liquation of the crusts is hindered by oxide films, and addition of a flux, such as zinc chloride or ammonium chloride, is beneficial. Further, a narrow deep kettle is better than a flat open kettle in avoiding excessive oxidation of zinc during the melting period.

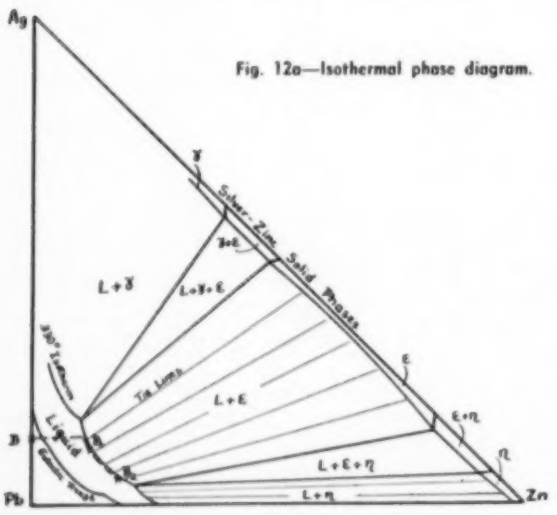

Fig. 12a—Isothermal phase diagram.

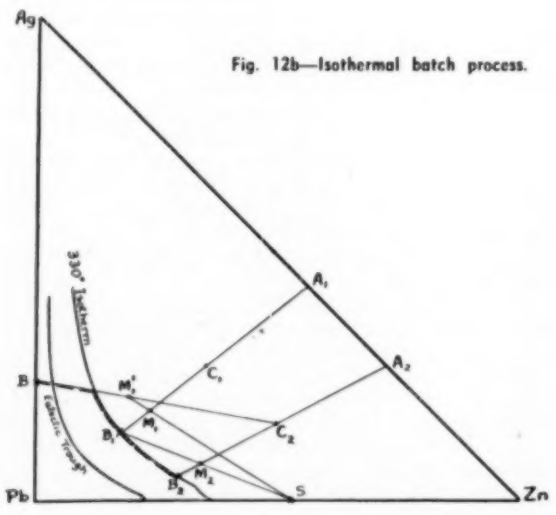

Fig. 12b—Isothermal batch process.

If the crust melting kettle is kept at a high temperature near the top and a low temperature near the bottom, rich alloy may be dipped from the top and completely desilverized lead tapped from the bottom, or more conveniently, siphoned out. This fact was recognized as a significant pointer to a method of refining lead for silver in a continuous flow.

The crust enrichment process was conducted at Port Pirie from 1923 to 1930 when continuous desilverizing commenced. It has also been employed at Kellogg since the late 1920's.[28]

A new publication[35] discloses the results, but not the procedure, of a crust enrichment process practiced in Brazil since 1951. From the description it appears that a process quite similar to that of Williams has been developed there independently.

Williams Continuous Desilverizing Process: It has already been pointed out that a batch of bullion can be completely desilverized with one zincing at a high temperature followed by cooling to nearly freezing.

It therefore follows that, if lead is saturated with zinc at the conjugate solution boundary, on cooling the lead low grade crusts will separate and desilverized lead will be produced when the temperature has fallen nearly to the freezing point.

In the Williams kettle[10-12, 28] lead containing silver is introduced at the top at high temperature (about 600° to 650°C) where a layer of molten zinc is maintained. At this temperature about 10 pct Zn can be absorbed by lead. The lead flows downward through the kettle, of which all but the top section is cooled by air draught. (The top section is fired to keep the zinc-rich layer molten.) As the temperature of the lead falls, the saturation limits of zinc and silver progressively fall, so that solid ϵ-phase Ag-Zn is continuously rejected. The cooling of the kettle is so arranged that at the bottom lead is very close to its freezing point, and thus its composition is at a point very close to the eutectic trough.

For practical reasons the lead near the bottom of the kettle enters a so-called siphon pipe which leads it upward again through the hotter zones of the kettle, so that it is finally discharged at a temperature of about 350°C, well above freezing point. Also, to ensure that the lead entering the siphon pipe is very close to its freezing point, a solid layer of lead is maintained on the bottom of the kettle. The thickness of the solid layer is measured periodically by a probe inside the siphon pipe extending to near the kettle bottom. If the solid layer decreases in thickness, more vigorous cooling is applied to the lower part of the kettle; if it increases, less. In this way, temperature control of a large body of lead (350 long tons) is maintained without adjustments.

The alteration in bullion composition during this operation can be studied in Fig. 13, which is schematic only. The initial composition (Pb-Ag only) is B. The bullion absorbs zinc at the top of the kettle, bringing its composition to B_1. The zinc layer absorbs lead and silver, bringing its composition to A_1. As bullion falls in temperature by moving downward in the kettle, its composition changes from B_1 to B_2. As it discharges from the kettle into the siphon pipe, its composition is B_2, lying nearly on the eutectic trough. The bullion thus behaves exactly as if in a batch one-stage zincing operation.

The crusts, which separate from bullion that changes from B_1 to B_2 while cooling, will now be considered. They rise to the surface of the lead where

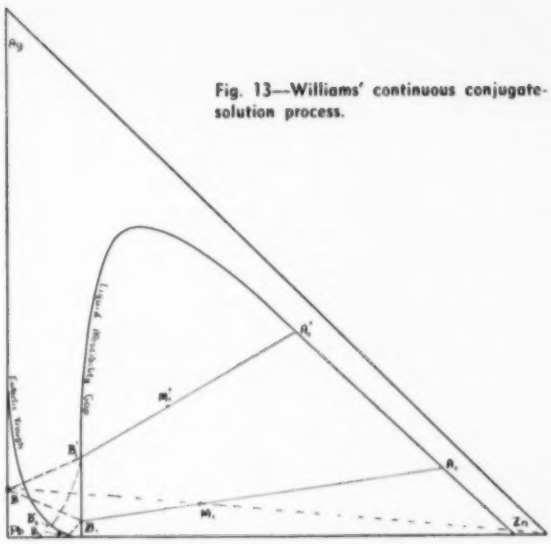

Fig. 13—Williams' continuous conjugate-solution process.

the temperature is sufficient to melt them, or at least to dissolve them in the zinc layer.

The adjacent portions of lead and zinc layers, which together had a total composition M_1, thus gradually become enriched in silver, and move in the direction of M'_1. This mixture is seen to be composed of a zinc-rich solution A'_1, and bullion B'_1. The bullion B'_1 when it reaches the bottom of the kettle will have a composition B'_2, higher in silver and lower in zinc than the lead which flowed out earlier. The melting point of A'_1 is at a higher temperature than the earlier alloy A_1. Thus the longer the lead flow is continued through the kettle, the greater is the theoretically expected silver content of desilverized lead, and the higher the melting point of the Zn-Ag alloy layer on top of the kettles. The process obviously cannot continue in this way indefinitely.

Port Pirie practice is to dip off the silver alloy layer when its appearance indicates that it is about to solidify. The desilverized lead at that stage still contains less than 0.0003 pct Ag, and in fact it has not been demonstrated that the silver content of desilverized lead rises throughout a cycle.

When the silver alloy has been dipped off, fresh zinc is added to the top of the kettle where it melts, and a new cycle commences, the alloy layer moving in composition from pure zinc to A_1 and then gradually to A'_1. The lead flow is not stopped while skimming the alloy or adding fresh zinc so that the lead flow is continuous, whereas removal of the alloy is discontinuous. Once a week the lead flow is stopped, the kettle sides heated to free adhering crusts which have not floated, and the kettle bottom cooled to nearly freezing before the lead flow is recommenced. The stoppage is normally for 8 hr.

In order to correct certain misconceptions of the limitations of this process, the author has previously published a short account.[29] For completion here, a statement of the advantages and disadvantages of the process as compared with other methods is included:

1—It is evident from the foregoing description that calculation of zinc additions is unnecessary. The input bullion silver content need not be known. The bullion may be very high or very low in silver, and long-term variations or short-term fluctuations have no effect whatsoever upon the operational con-

trol. The silver alloy is dipped off when its silver content is high enough to cause incipient freezing. If the silver content of input bullion increases, silver alloy must be dipped more frequently, but advance information by assaying is not needed. The silver and zinc contents of bullion in the kettle bear no relation whatever to the silver content of bullion entering at that instant. They are determined by the temperature at the top of the kettle, and the stage reached in a cycle; thus toward the end of a cycle, the lead entering the kettle may actually absorb silver from the alloy layer (B to B'_1).

2—The bullion flow to the kettle may vary widely without adversely affecting operations, provided only that the temperature at the bottom is controlled by increase or decrease of draught cooling, the necessity for which is indicated by the thickness of the solid layer at the bottom.

3—It is not necessary to press the alloy dipped as it is already liquated, although incompletely, because oxide particles prevent complete separation of lead. Thus in practice the alloy dipped contains 15 pct Ag, 20 pct Pb, and 65 pct Zn. The Ag/Zn ratio is lower than that obtained in a two-stage batch process. To obtain a ratio of one-half, comparable with the latter, a top temperature of 610°C theoretically suffices. In practice this temperature is insufficient to maintain the alloy layer liquid if the silver content exceeds the figure quoted.

4—The whole of the bullion treated must be heated to a high temperature in order first, to liquate the crusts formed, and second, to ensure sufficient solution of zinc for complete desilverizing. Maintaining a layer of molten zinc-rich alloy at high temperature entails severe corrosion of the cast-iron kettle top, so that replacement costs are proportional to time, and not to the tonnage treated. Thus the economic attractiveness of the process is less the lower the treatment rate. If a material could be found which is of high thermal conductivity and resistant to molten zinc corrosion, this disadvantage would disappear. There is no fundamental reason why infinitesimally small bullion flows, with silver contents varying from 0 to 100 pct, should not be treated by this process.

Proposal for a New Low Temperature Continuous Desilverizing Process: This process was conceived to overcome the disadvantages of high temperature incurred by Williams' continuous process. Savings in fuel and maintenance costs should result from its development, particularly for small producers. Advantage could then be taken of the greatly superior continuous processes for the other refining steps.

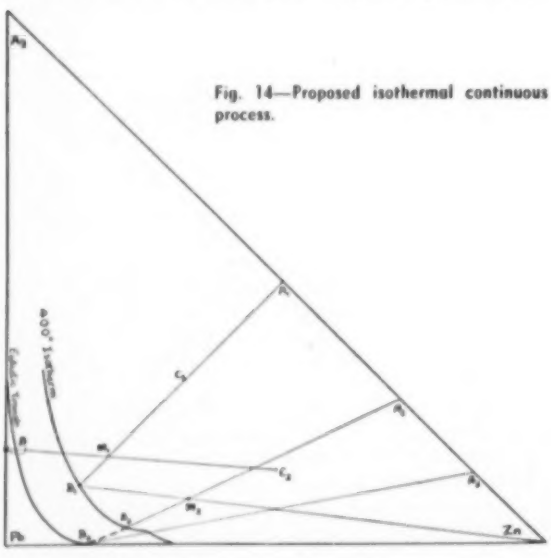

Fig. 14—Proposed isothermal continuous process.

The process is essentially the same as that described above as a low-temperature isothermal process, conducted continuously instead of in batches. The process may be conducted in one stage, yielding a low grade silver alloy on liquation, or preferably in two stages, yielding a very high grade silver alloy on liquation.

Bullion flows continuously into a kettle where the second crusts are stirred in. The outflowing mixture is separated in another kettle into first crusts and bullion which flows on to the second stirring kettle. Here the zinc addition is made and the resulting mixture flows to a separating and cooling kettle similar to the bottom section of Williams' kettle. The second crusts separating here are returned to the first stage.

The new zinc additions to the second stirring kettle must always be sufficient to produce the required low silver value in desilverized lead, and thus depend upon the input bullion silver percentage. If the highest input silver value is known, the amount of zinc required to desilverize bullion of this composition could be added continuously, the only disadvantage of this procedure being that the Ag/Zn ratio would not be at all times the highest possible. It would approach one-half and compare favorably with that obtained in good batch processing, if the input silver value does not vary greatly.

The method is illustrated in Fig. 14 where it is assumed that the operating temperature is 400°C for both stages, although operation is possible at all temperatures from about 330°C upward. Above 420°C, of course, the zinc added to the second stirring stage would melt, and some corrosion of the kettle and stirrer would result. Whether this would be of serious concern is a matter for practical trial.

The bullion B entering the system is stirred with the second crust C_2, yielding mixture M_1, which at equilibrium at 400°C comprises bullion B_1 and the ϵ-phase Ag-Zn alloy A_1. The first crust skimmed C_1 would give alloy A_1 and bullion B_1 on complete liquation.

Bullion entering the second stage B_1 is mixed with pure zinc, yielding mixture M_2, which at equilibrium at 400°C comprises bullion B_2 and the ϵ-phase Ag-Zn alloy A_2. In the final cooling kettle the bullion changes in composition from B_2 to B_3, the final desilverized lead, and rejects ϵ-phase Ag-Zn alloy varying in composition from A_2 to A_3. The resulting crust skimmed from the final cooling kettle is C_2 (lying between B_2A_2 and B_3A_3) which is returned to the first stirring stage.

The plant required comprises four kettles operated at a constant temperature and two stirrers operated continuously. The zinc addition should be proportioned to the silver input and lead throughout, and can be readily calculated from the appropriate cooling curves. A liquating kettle or furnace is required for separating the lead entrained in the first crust, otherwise retorting and cupellation costs would be rather high, as the crusts from 0.184 pct Ag input bullion (60 oz per long ton) would contain approximately 10 pct Ag, 20 pct Zn, and 70 pct Pb.

Degolding by The Parkes Process

At one time it was general practice at lead refineries using the Parkes process to produce a so-

called gold skim before desilverizing proper. Practically all the gold and copper, together with a fraction of the silver, are removed by adding a relatively small amount (0.3 pct) of zinc to the bullion.

The explanation invariably given for this phenomenon is that "zinc has a greater affinity for gold and copper than for silver." It is difficult to improve upon this explanation, unless the more sophisticated expression is preferred: that the free energies of formation of Zn-Cu and Zn-Au phases are greater than those of Zn-Ag phases. Figures are quoted for phases of unspecified composition by Henglein and Nowotny,[31] namely, −5.2, −2.8, and −2.2 kcal for Au-Zn, Cu-Zn, and Ag-Zn, respectively. The differences are great enough to ensure the removal of practically all the gold and copper with as little as 13 pct of the silver in practice.[28] The gold to silver ratio in the gold skim was increased by recycling crusts from treated batches to succeeding batches. In the case cited[28] the Bunker Hill smelter adopted the Port Pirie practice of recycling the gold crusts through five kettles.

This practice of producing a small amount of crust containing all the gold and copper, and a larger amount of silver crust which is free from gold and copper afforded two advantages. First, the amount of doré bullion to be parted was greatly reduced; and second, the cupellation of retort bullion is much more rapid in the absence of copper. The practice has fallen into disuse, presumably because the extra treatment stage in the main line of lead production is more expensive than extra work involved in the byproduct treatment section for cupelling and parting. No doubt improved removal of copper prior to softening was also a big influence; it is stated to be the main factor responsible for the elimination of degolding at Port Pirie.[28]

Recently, however, the process was revived in a new application. Although degolding of softened bullion may be uneconomical, in some circumstances degolding retort bullion offers economic advantage.

Frayne[32] developed and introduced such a degolding practice at Namtu, Burma, in 1934. From lead containing 0.25 parts Au per 1000 parts Ag, he produced two silver products, one containing 0 to 0.03, the other up to 3.5, parts gold per 1000. The publication of this achievement provoked a good deal of adverse criticism[33] but the fact remains that Frayne pioneered a new development which may well prove of value to all lead refineries. The primitive methods which Frayne used seem certain to be improved.

Fig. 15 shows two alternative flowsheets by which the process may be carried out. Fig. 15a is the simplest possible, but would not provide the degree of enrichment afforded by Fig. 15b unless very accurate thermal and analytical controls were constantly maintained.

The process will not be discussed in conjunction with a phase diagram, since the quinary system Ag-Au-Cu-Pb-Zn cannot be represented readily. It is sufficient to state that when limited zinc additions are made to bullion, all the gold and copper may be removed, together with only a portion of the silver; also that the Au/Ag ratio of any crust formed is higher the higher the temperature of its formation.

Following the flowsheet of Fig. 15a, it can be seen that the retorting of silver alloy must be stopped while there is sufficient zinc remaining to permit conducting a Parkes process on the retort bullion. Alternatively, new alloy could be added to provide the zinc required. The retort bullion is cooled until

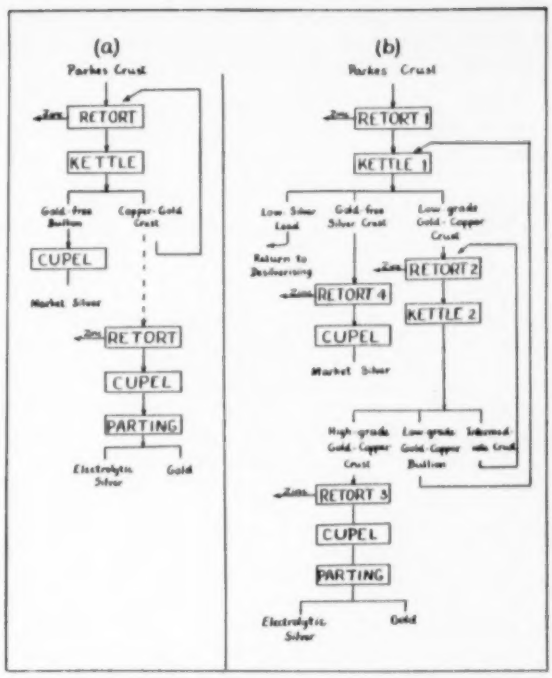

Fig. 15—Flowsheets, *a* and *b*, for degolding of retort bullion.

all the copper and gold have been precipitated as a crust, which is removed, leaving copper-free, gold-free bullion which may be cupelled to yield market silver. The Cu-Au crust may be recycled through the retort until a very high concentration of copper and gold is reached when the crust is separately retorted, cupelled, and parted.

In the flowsheet of Fig. 15b two additional features are provided. The copper-free, gold-free retort bullion is allowed to cool, the crusts are skimmed, and the remaining low silver lead is returned to the lead desilverizing section. This lead is not lost in cupellation, but returned directly to the refinery. The rich silver crusts are cupelled to give market silver after retorting to recover zinc, if present in sufficient quantity to warrant recovery.

The Cu-Au crusts are not recycled through No. 1 retort, but are retorted separately and enriched by recycling through this second retort and cooling kettle to any desired extent. The final very high grade Au-Cu crusts, which are a very small fraction of the original retort bullion quantity, are removed, retorted if warranted, and then cupelled and parted.

The economic evaluation of such a treatment depends upon the balance between power and labor saved in the parting plant plus lead returned directly to the refinery, as against slightly increased retorting costs, the labor cost of the degolding process, and zinc lost by oxidation when recycling crusts, although there could be a net zinc gain if less zinc were lost in cupellation.

It is not possible to quote the exact temperatures required for all the combinations of silver, gold, copper, zinc, and lead encountered in retort bullions in practice. It suffices to say that Frayne operated his degolding process between 550° and 480°C by diluting retort bullion with lead to give baths of 3000 oz per ton Ag in both degolding and gold crust enrichment kettles. The process can be operated at higher temperatures in bullion baths of much higher

silver content, the author having used baths of 12,000 to 13,000 oz per ton Ag experimentally at temperatures around 600°C.

Conclusion

Published work which has contributed to the present day knowledge of the ternary system Ag-Pb-Zn has been reviewed with special emphasis on the section of the diagram (the lead corner) concerned in the practical desilverizing processes. The field of practical interest is still imperfectly known. Although the course of the eutectic trough and the isotherms on the liquidus surface between the eutectic trough and miscibility gap boundary are quite well established, there is need for a precise determination of the compositions of Ag-Zn phases in equilibrium with liquid lead, containing silver and zinc in solution, for a range of composition and temperature. There is need also for confirmation of the position of the peritectic transformation point between ϵ and η Ag-Zn phases, liquid, and solid lead, which this paper has assumed to coincide with Johannsen and Lange-Eicholz's supposed ternary eutectic point.

The course of practical desilverizing processes has been demonstrated on schematic phase diagrams. The desilverizing of lead bullions in practice is carried out, not in the miscibility gap between two liquid phases, but in a similar gap between the solid ϵ-phase (Ag-Zn) and liquid lead-rich solution. When very low silver contents in desilverized lead are obtained (below 0.0005 pct Ag), the η-phase (Ag-Zn) is rejected from the lead, at least in the last stages of cooling.

A fundamentally sound process for continuous desilverizing of lead at a low temperature has been proposed. No radically new engineering techniques, but development of suitable plant design, would be necessary. The process should be more economical than the Williams method for small producers, but has the comparative disadvantage that calculation of zinc addition, based on input bullion tonnage and silver analysis, would still be required.

A recent innovation, that of degolding and decoppering retort bullion by Parkes treatment, has been briefly discussed in qualitative fashion. A few laboratory tests on any particular grade of retort bullion would show whether the process has economic value for the plant concerned.

Acknowledgments

The author is indebted to many colleagues at Port Pirie and the University of Melbourne for discussion and helpful criticism of the draft of this paper. The permission of the Management of The Broken Hill Associated Smelters Pty., Ltd. to present this paper is gratefully acknowledged.

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* References 2, 3, 5, and 8 have not been consulted in the original by the author.

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aime NEWS

Gordon Research Conference To Be Held

The 1954 Gordon Research Conference on the Chemistry and Physics of Metals will be held from Aug. 16 to 20, 1954 at the New Hampton School, New Hampton, N. H. The Gordon Research Conferences are sponsored by the American Assn. for the Advancement of Science and were established to stimulate research in universities, research foundations, and industrial laboratories.

Papers are to be presented on the mechanical properties of metals.

Requests for additional information can be addressed to W. George Parks, director, Gorden Research Conferences, Colby Junior College, New London, N. H.

Back Issues Requested

Institute headquarters will pay 50¢ for back issues of the following magazines received: MINING ENGINEERING; October 1953, November 1953, January 1954, February 1954. JOURNAL OF METALS; August 1953, March 1954, April 1954.

Division Officers Nominated for 1955

Institute of Metals Div.

Chairman: J. S. Smart, Jr., American Smelting & Refining Co.; Senior Vice-Chairman, C. S. Barrett, University of Chicago; Vice-Chairman, W. R. Hibbard, consultant; Secretary, E. O. Kirkendall, AIME. Executive Committee (three years): J. Nielsen, New York University; R. S. Busk, Dow Chemical Co.; R. Maddin, Johns-Hopkins University.

Iron and Steel Div.

Chairman: A. W. Thornton, U. S. Steel Corp.; Vice-Chairmen (one year): S. Marshall, U. S. Steel Corp.; G. B. McMeans, Kaiser Steel Corp.; H. B. Emerick, Jones & Laughlin Steel Corp. Executive Committee (three years): V. W. Jones, Armco Steel Corp.; L. R. Berner, Inland Steel Co.; L. W. Moore, Crucible Steel Co.

Extractive Metallurgy Div.

Chairman: A. W. Schlechten, Missouri School of Mines; Chairman-Elect, J. C. Kinnear, Jr., Kennecott Copper Corp.; Secretary, E. O. Kirkendall, AIME. Executive Committee (three years): J. D. Sullivan, Battelle Memorial Institute; T. D. Jones, American Smelting & Refining Co.

Mineral Industry Education Div.

Chairman: J. R. Van Pelt, Jr., Montana School of Mines; Vice-Chairman, J. W. Stewart, University of Alabama; Secretary, O. C. Shepard, Stamford University. Executive Committee (three years): Charles Meyer, University of California; J. W. Vanderwilt, Colorado School of Mines; R. D. Parks, Massachusetts Institute of Technology.

Bylaw Revision Passed

The revision in Art. XI, Sec. 3, of the AIME bylaws, published in the April issues of the three Institute journals, was approved by the Board at its May 19 meeting. In short, the amendment removes the limitation of $400 as the maximum grant that may be given to a Local Section in any year.

The aid given to Local Sections who request it in 1954 consists of 50¢ per member on Jan. 1, 1954; half of the initiation fee received from all new members; and the first class railroad fare to and from the Annual Meeting for the Section Delegate. Aid to be granted in 1955 has not yet been decided upon in any of these three respects. Under the new plan, Local Sections should receive approximately $25,000 in 1954 if they get as many new members as last year, compared with $19,429 in 1953.

Amend Nominating Method for Officers

Certain changes in the nominating procedure for AIME officers and directors were approved by the Board at its May 19 meeting. Briefly, the procedure in future will be as follows:

The Nominating Committee will consist of 18 members, none of whom shall be a director or director-elect. Eleven of these shall be named by the Council of Section Delegates, two by the Mining Branch Council, and one each by the Petroleum and Metals Branch Councils. The other three, including the chairman, shall be named by the president. The committee will meet during the week of the Annual Meeting, and will select a slate of nine names, one for President-elect, two for Vice-Presidents, and six others for directors, all for three-year terms except the President-elect, who will successively become President and Past President. The committee will make its selections to the end that the directors come from different parts of the country and are representative of the various Divisions and Branches of the Institute. Twelve votes out of the 18 are required to name a candidate for the slate, the chairman having a vote. If no candidate receives 12 or more votes for the respective nine offices at the meeting of the committee, a letter ballot or ballots shall be conducted among the members of the committee within a 60-day period. If the letter ballots fail to provide nine candidates each with 12 or more votes, then the two highest candidates shall be submitted to all members of the Institute for letter ballot. Selections thus made will be published in the July issues of the AIME journals, with biographical sketches. Any 25 Members or Associate Members can then petition to have one or more additional names put on the ballot. If no such petitions are received by September 1, the nominations of the committee are accepted by the Board without a letter ballot.

The principal change just made by the Board is that a two thirds majority instead of a simple majority is necessary for selection of the candidates by the Nominating Committee.

ANNUAL MEETING PAPER DEADLINE

Sept. 15, 1954 is the deadline for all Institute of Metals Div. papers for the 1955 Chicago Annual Meeting and the deadline for all Iron & Steel Div., and Extractive Metallurgy Div. papers to be preprinted. Papers received by this deadline but requiring revision may not be processed in time to permit scheduling for this meeting.

Petroleum Executive Heads 1955 AIME Nominations

For President

C. E. Reistle, Jr., member of the Board of Directors of Humble Oil & Refining Co. in charge of the production dept., has been nominated for the Presidency of AIME in 1956. Born in 1901 in Denver, Colo., Mr. Reistle was graduated from the University of Oklahoma in 1922 with a B.S. degree in chemical engineering. He began his career in the oil industry by working as a roustabout for the Carter Oil Co. during summer vacations while attending college. Soon after his graduation Mr. Reistle joined the U. S. Bureau of Mines in Bartlesville, Okla., as a junior petroleum chemist. While with the Bureau he visited and worked in many of the major oil fields in the country. During this time he published a number of articles on oil field brines, paraffin problems in the production of crude oil, and the operation of flowing wells. In 1933 Mr. Reistle left the Bureau of Mines to become chairman of the East Texas Engineering Assn., a cooperative organization formed to obtain engineering facts on the efficient operation of the East Texas field. He was in charge of gathering these facts, with the aid of the companies operating in the field. His efforts played an important part in establishing efficient production rates and practices for the field. In August 1936 Mr. Reistle joined Humble Oil & Refining Co. as engineer-in-charge of the petroleum engineering div. and in 1940 was advanced to the head of the division. He remained chief petroleum engineer until February 1945, when he was made general superintendent of the Production dept., and in August of the same year he became manager of production operations. Mr. Reistle was elected to the board of directors of the Humble Co. in 1948. During World War II he was technical advisor for District III of the Petroleum Administration for War. He was also national vice-chairman of the Oil Industry Advisory Committee for OPA.

C. E. REISTLE, JR.

For Vice-Presidents

Walter A. Dean, Cleveland works chief metallurgist of the Aluminum Co. of America, has been nominated for Vice-President and Director of AIME. Mr. Dean attended Cooper Union Institute of Technology in New York City and received his B.S. degree in chemical engineering from there in 1926. He continued his studies at Rensselaer Polytechnic Institute receiving his M.S. degree in 1927 and a Ph.D. in 1929. While there he was elected to Sigma XI. On graduating from Rensselaer he accepted employment at the research laboratories of the Aluminum Co. of America located at New Kensington, Pa. In 1931 he was transferred to the research laboratories at Cleveland where he remained until 1945. He was named assistant manager of the permanent mold plant at Cleveland in 1945 and in 1949 became the Cleveland works chief metallurgist, the position he holds to this date. His work at the research laboratories was concerned largely with the development of alloys and processes. He has written articles on machineability of aluminum and is the author or co-author of approximately 90 patents covering alloy and process developments.

William W. Mein, Jr., who has been nominated as a Vice-President and Director of AIME, is president of Calaveras Cement Co., Bishop Oil Co., and Clear Lake Water Co. all of San Francisco. Mr. Mein is a graduate of Harvard College, Class of 1932, with an A.B. degree in engineering sciences. Shortly after graduation he started with Calaveras Cement Co. as field and office engineer at the cement plant in San Andreas, Calif. Following this position, he spent two years as petroleum production engineer and oil geologist for Bishop Oil Co. at various oil field locations in Texas and Oklahoma. Resuming his employment with Calaveras Cement Co. in San Francisco, Mr. Mein progressed from office engineer to purchasing agent to assistant vice-president to vice-president. In April 1943 he was also elected vice-president of Bishop Oil Co. In June 1947 he became president of Clear Lake Water Co. Mr. Mein was elected president of Calaveras Cement Co. in April 1951 and president of Bishop Oil Co. in May 1953. From 1944 to 1945, he was appointed by Governor Earl Warren to the California State Mining Board and became its secretary. He has recently served on the National Minerals Advisory Council of the Dept. of the Interior and the Natural Resources Committee of the U. S. Chamber of Commerce. In order to devote more time to his family and work he recently resigned from the Woodside School Board after being elected to three terms. Besides his directorships in Calaveras and Bishop, he is also on the board of Commonwealth Investment Co. and Clear Lake Water Co.

W. A. DEAN

W. W. MEIN, JR.

Grover J. Holt, Chairman of the Nominating Committee, has announced nominations for President-Elect, Vice-Presidents, and Directors for 1955. As provided in Art. IX, Sec. 2, of the bylaws, 25 Members or Associate Members may sign and transmit to the Secretary's office prior to Sept. 1 "any complete or partial ticket of nominees," should they wish other candidates to be considered.

In such instance, a letter ballot will be forwarded to all Members in good standing in the United States, Canada, and Mexico, tabulating both the official ballot and any supplementary nominations. If no supplementary nominations are thus received, no letter ballot will be printed, and nominees on the official ballot shall be declared duly elected at the meeting of the Board of Directors or the Executive Committee in November.

For Directors

Richard Whitsett French, Jr., vice-president of production for the Standard Oil Co. (Ohio), has been nominated as a Director of AIME. Born in Colorado Springs, Colo., Mr. French attended the Universities of Delaware, Southern California, and Harvard. He began his career with the General Electric Co. While at GE he worked on marine and aeronautical engineering. Later Mr. French joined the U. S. Air Corps as a flying cadet. He served at Kelly Field as an armament officer and assistant engineering officer. He held the rank of second lieutenant when he was discharged. In 1933 he joined the Continental Co., Ventura, Calif. He worked his way up to field engineer and in 1936 was promoted to production engineer in the Los Angeles office. For the next seven years he directed all of Continental's petroleum and production engineering in California. In 1943 he was moved to Ponca City as assistant manager of the production dept. While here his work covered all phases of engineering and production as well as administrative duties. Mr. French remained with this firm until 1947, when he accepted a position with the Sohio Petroleum Co., as chief engineer and assistant to the vice-president in charge of production. In 1948 he was named vice-president. Mr. French was appointed vice-president and general manager of the production dept. for Sohio in 1953. Since April 1954 he has been vice-president of production for the Standard Oil Co. (Ohio).

D. C. Helms has been nominated Director of AIME. Mr. Helms has been associated with the Lehigh Navigation Coal Co. since 1909. Prior to that he had worked at the Brooklyn Navy Yard and American Bridge Co. For a short time he was associated with the St. Clair Coal Co. and Philadelphia & Reading Coal & Iron Co. His first position with Lehigh was as a chairman and draftsman. He became division engineer, assistant mining engineer, and stripping superintendent, respectively. In 1918 he was named superintendent of the Alliance Coal Mining Co. During the years 1918 and 1938 he was superintendent of several divisions for the firm. Mr. Helms was appointed mining engineer and cost supervisor for the company in 1938. From 1944 to 1947 he was production manager and was then appointed general manager. He was vice-president for two years, prior to his retirement.

R. W. FRENCH, JR.

D. C. HELMS

A. J. HERZIG

H. C. PYLE

Alvin J. Herzig, recently elected vice-president, research, Climax Molybdenum Co., has been nominated as a Director of AIME. Mr. Herzig was born at Toledo, Ohio, and graduated from the University of Michigan. In 1926 he was employed by the National Supply Co., Toledo, where specialization in physical metallurgy began. He returned to the University of Michigan to conduct research work on brass and work for an advanced degree. He received his M.S.E. in 1931. After receiving this degree he joined the staff of the Climax Molybdenum Co., as chief metallurgist. In 1943 Mr. Herzig was made vice-president in charge of the laboratory. Mr. Herzig was elected to the presidency of Climax Molybdenum Co. of Michigan in 1949.

Howard C. Pyle, president of Monterey Oil Co. since November 1951, has been nominated as a Director of AIME. He graduated from the University of California in 1926 with a bachelor of science degree and subsequently received a master of science degree from the University of Southern California. During the first two years of development at Signal Hill, Mr. Pyle worked as a roustabout, pipe fitter, and roughneck. He was located in Venezuela as a geologist for the Union Oil Co. in 1927. In 1928 Mr. Pyle was transferred to California and in 1933 was promoted to petroleum engineer and chief production engineer. In January 1943 he entered the Army Corps of Engineers as a captain and was assigned to duty in Washington, D. C. He was later transferred to England to the general staff of General Eisenhower. During the Normandy invasion he served on Field Marshall Montgomery's staff. In October 1944 Mr. Pyle was promoted to lieutenant colonel and served as deputy chief of the G-4 petroleum branch, communications zone. He was discharged in 1945. At this time he was named vice-president, Bank of America, a position he held until November 1947. For three years, Mr. Pyle was president of Continental Consolidated Corp. and Continental Corp. Prior to joining Monterey he was a consulting petroleum engineer in Los Angeles.

E. H. Rose, who has been nominated a Director of AIME for a three-year term, is a Member of the Southeast Local Section, AIME. Born at Kinsley, Kan., Mr. Rose is a graduate of the University of Kansas. He began his career as a mill superintendent with the Patino Mines, in Bolivia. From 1928 to 1930 he held the same position with the Moctezuma Copper Co. in Mexico. For six years Mr. Rose was assistant mill superintendent for the International Nickel Co. of Canada. In 1936 he was promoted to mill superintendent, a position he held until 1945. He was named consulting metallurgist for the Copper Range Co. on the White Pine project in Michigan during 1946. He joined the U. S. Steel Corp., Tennessee Coal & Iron Div. in 1947 and until recently was research engineer. He has been a member of the raw materials advisory committee, U. S. Atomic Energy Commission since 1949.

E. H. ROSE

J. W. VANDERWILT

John W. Vanderwilt, president of the Colorado School of Mines, Golden, has been nominated as a Director of AIME. Dr. Vanderwilt is a native of Oskaloosa, Iowa. He attended the University of Michigan and Harvard Graduate School. Following graduation from the University of Michigan, he was an instructor in geology at the University of Colorado, Boulder, Colo. For one year he was located in Washington, D. C., as a junior geologist with the U. S. Geological Survey. Dr. Vanderwilt returned to the University of Colorado as an assistant professor of mineralogy in 1928. From 1929 to 1934 he was an assistant geologist for the USGS. In 1934 Dr. Vanderwilt became a consulting mining geologist. In 1947 he returned to Colorado, as a member of the board of trustees, Colorado School of Mines and in 1950 was named president. In 1953 he became a member of the committee on mineral research, National Science Foundation.

One of the memorable events in the Pacific Northwest Conference was the Grand Banquet held Friday evening in the Rose Bowl of the Hotel Multnomah.

Business and Pleasure Mark AIME 1954 Pacific Northwest Metals and Minerals Conference

BONNEVILLE Power Administrator Dr. W. A. Pearl told some 275 delegates to the AIME Pacific Northwest Metals and Minerals Conference held April 29 to May 1 in Portland that the greatest growth for the region's metallurgical industries, in spite of recent rapid strides, lay in the future.

Dr. Pearl spoke at the Metals Branch luncheon, stating the development of low grade ores in the Northwest depends on improving electrolytic processes and obtaining increased supplies of hydroelectric power. He expressed confidence that increased power supplies would be available when needed.

Delegates to the three-day convention, held in Portland's Multnomah Hotel, pointed with pride to the increasing attendance at the annual affairs. Founded shortly after World War II, the conferences have mushroomed from one-day (two sessions) industrial minerals meetings to the present three-day (11 sessions) gatherings. Next year's meeting will be in Spokane with the Columbia Section handling the details.

General Chairman of the Conference was Frank X. Cappa who also served as toastmaster for the banquet.

General Chairman Frank Cappa, Aluminum Co. of America, Vancouver, Wash., called the recent conference "extremely successful" and forecasted an even better turnout next year.

Another featured speaker was Felix E. Wormser, Assistant Secretary for Mineral Resources, Dept. of Interior. He told guests at the grand banquet that he disagreed with those who saw the U. S. rapidly becoming a have-not nation in minerals. He said that up to the present day only the high grade ores at or near the surface had been exploited.

Mr. Wormser stated his belief that much more ore exists without surface indications and that the work of the U. S. Geological Survey would assist in discovering these unexposed orebodies. The work of the U. S. Bureau of Mines, he added, would lead to better techniques so the extensive low grade deposits now known could be utilized.

Harold E. Lee and Donald Ingvoldstad, both of Bunker Hill Smelter, were co-Chairmen of the extractive metallurgy session. The iron and steel and physical metallurgy programs were chairmanned by G. R. Heffernan, Western Canada Steel, Ltd., and S. F. Maddigan, Kaiser Aluminum & Chemical Corp., respectively.

F. E. Wormser was the principal speaker at the Banquet. From left to right are: Mrs. Marble; E. M. Marble, AIME Director; Mrs. McNaughton; R. R. McNaughton, Chairman of Metals Branch; and Mr. Wormser.

E. M. Marble introduced W. A. Pearl, Bonneville Power Administration, the principal speaker at the Metals Branch Luncheon Friday noon. From left to right are: E. M. Marble, Dr. Pearl, and Mr. Peterson, mayor of Portland.

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Personals

R. L. Bobo was appointed Great Lakes regional manager for Federal Electric Products Co., and subsidiary, Pacific Electric Mfg. Corp.

Thomas C. Baker, Jr., American Manganese Steel Div., American Brake Shoe Co., has been transferred from Cleveland to Chicago. He is now working in the capacity of Chicago district sales manager.

N. W. Richardson has been appointed representative in Colorado, New Mexico, and west Texas by the Geo. P. Reintjes Co.

Harry K. Taylor was named assistant vice-president in charge of operations for Jessop Steel Co. Mr. Taylor joined Jessop in 1944.

Thomas C. Ford, Pittsburgh Metallurgical Co., Niagara Falls, N. Y., has joined the Ford Tool & Carbide Co., Birmingham, Ala.

Rex F. Pearce is no longer associated with South African Cyanamid Pty. Ltd., Johannesburg. He has accepted a position with Sherritt-Gordon Mines Ltd., chemical metallurgical div., at the new nickel refinery, Fort Saskatchewan, Alberta.

Bernard N. Zimmer, vice-president and director of the American Metal Co., Ltd., will retire as vice-president. He has been with the company for 40 years. Mr. Zimmer will continue as a member of the board of directors.

John J. O'Connor, Jr., was appointed director of public relations at Polytechnic Institute of Brooklyn.

James M. Mead has been elected a vice-president and director of Joseph T. Ryerson & Son, Inc., Chicago.

Robert W. Norton has been appointed vice-president and **Myron B. Diggin** named assistant vice-president by the Hanson-Van Winkle-Munning Co.

Joel Hunter was elected president of Crucible Steel Co., succeeding **William H. Colvin.** Mr. Colvin will continue as a member of the executive committee and board of directors.

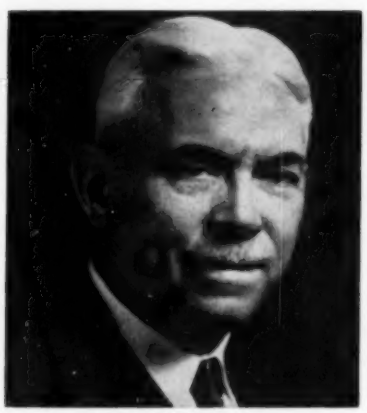
Theodore H. Harley has joined the Haws Refractories Co., Johnstown, Pa., as ceramic engineer. He had been assistant to mason superintendent, Ford Motor Co., steel div., Dearborn, Mich.

Maj. Gen. William H. Porter has been appointed a member of the board of directors of Horizons Titanium. General Porter, active consultant to American Cyanamid Corp., served as president of Chemical Construction Corp. from 1947 to 1952 and in 1953 was elected as chairman of the board. He retired in 1953 when he became active consultant.

F. L. LaQUE

O. B. J. FRASER

D. J. REESE



T. H. WICKENDEN

Frank L. LaQue has been elected vice-president of the International Nickel Co., Inc., and manager of the development and research div. Mr. LaQue succeeds **Thomas H. Wickenden** who has attained retirement but will continue to serve as a consultant on special projects. **O. B. J. Fraser** and **Donald J. Reese** will serve as assistant managers of the div. Mr. Fraser is a vice-president of AIME.

Vernon Swan, for the past 3½ years ingot products supervisor, Reynolds Metals Co., sales headquarters at Louisville, has been made manager of plant 5, located on Camp Ground Road, Louisville.

E. B. Jones has been appointed chief engineer in charge of estimating and engineering for the Lindberg Engineering Co. The position is in the newly formed field erected equipment div. Mr. Jones recently resigned from the Continental Industrial Engineers, Inc.

William C. Woodward has been appointed manager of forging sales for Aluminum Co. of America, Cleveland. Mr. Woodward had served as assistant manager of forging sales.

Alan D. Dauch has been appointed vice-president of sales, Salem-Brosius, Inc., Pittsburgh, and its recently acquired subsidiary, George J. Hagan Co., Pittsburgh. **John W. Holzwarth** is manager of sales of heavy heating furnaces for steel and nonferrous metals producing and fabricating industries and of all mechanical products. **Charles H. Schwerin** was named manager of sales of rotary hearth furnaces and all heat treating furnace equipment.

Paul E. Weingart, American Metal Co., Ltd., was elected president of the Metal Powder Assn. **William E. Cairnes,** president, Radio Cores, Inc., was elected chairman of the board; and **Robert L. Ziegfeld,** reelected secretary-treasurer.

M. E. GOETZ

Mowry E. Goetz, manager of operations, Cleveland district, Republic Steel Corp., received the David Ford McFarland Award for achievement in metallurgy. The award is granted by the Penn State Chapter of the ASM.

Neal F. Harmon, Edwin M. Kenefake, and **James D. Helm,** have been named sales managers in the communications equipment unit of the General Electric Co., commercial equipment dept., Syracuse, N. Y.

James B. Hatch has joined the metal chemicals dept., American Cyanamid Co., industrial chemicals div. Prior to joining the firm in April 1954 as sales trainee, Mr. Hatch has been associated with Sylvania Electric Products, Inc.

G. C. DAVIS, JR.

George C. Davis, Jr., has been named technical sales manager of the new technical sales dept. of Kaiser Chemical div., Kaiser Aluminum & Chemical Corp., Oakland, Calif. Mr. Davis has been assistant general sales manager for the company until recently.

Ryukiti Robert Hasiguti was recently promoted to full professor at the University of Tokyo, dept. of metallurgy, faculty of engineering.

C. H. Ramsden, formerly president of Pacific Coast Engineering Co., was elected chairman of the board. He has served as head of the firm since 1939.

Franklin H. Bivins has been named general manager of the newly created Vitro Rare Metals Co., a div. of Vitro Corp. of America.

Winton E. Platte, formerly with Apex Smelting Co., Chicago, has joined the National Metallurgical Corp., Springfield, Ore., as works manager.

William H. Rowand, vice-president of Babcock & Wilcox Co., New York, was awarded the Newcomen Medal by the Franklin Institute.

Obituaries

An Appreciation of

Dr. Wendell F. Hess

Wendell F. Hess, 51, director of research at Rensselaer Polytechnic Institute, Troy, N. Y., and head of the dept. of metallurgical engineering, died suddenly April 21. He was one of the world's leading authorities on scientific aspects of welding.

Dr. Hess' death was termed a "great shock and tremendous loss to the Institute and his field of study" by Rensselaer officials.

Born Jan. 1, 1903, the son of the late Wendell Jr., and Anna M. Beiermeister Hess, Dr. Hess was educated in the Troy public school system. He was graduated from School 16 and Troy High School. He was graduated from R.P.I. with a bachelor's degree in electrical engineering in 1925. In 1928 he received a Doctor of Engineering degree from the Institute.

He became a member of the R.P.I. faculty in 1928—the beginning of a relationship that was to last until his death. He was promoted from instructor to assistant professor in 1930 and served in this capacity until 1938. From 1938 to 1945 Dr. Hess was an associate professor and head of the welding laboratory, before his promotion to full professor in the latter year.

Dr. Hess was appointed head of the dept. of metallurgical engineering and Robert W. Hunt professor of metallurgical engineering in 1947, and in 1953 was named to the position of director of research for the Institute.

An internationally recognized figure in the welding field, the teacher-researcher had written more than 50 scientific papers in the field of electric arc and resistance welding of steels, aluminum, magnesium and nickel alloys. Some of his most comprehensive research studies were on-spot welding of hardenable alloy steels; measurement of arc weld cooling rates, and flash welding of aluminum alloys.

The awards he received for his work were numerous. He received among others: The Lincoln Gold Medal of the American Welding Society in 1944; the University Award of the Resistance Welder Mfrs. Assn. in 1944, 1945, 1947 and first and second prizes in 1948; the American Iron & Steel Institute Medal in 1944, and the John Price Wetherill Medal of the Franklin Institute in 1948.

Dr. Hess was active in many professional societies and organizations. He was a member of Sigma Xi and Tau Beta Pi Fraternities, a past national president and former board of directors' member of the American Welding Society, a fellow of the AIEE, and a member of the American Society for Engineering Education, ASTM, ASM, AIME, and Franklin Institute.

The R.P.I. professor was also a research consultant for Babcock & Wilcox Co. Previously he had been retained as a research consultant by American Bureau of Shipping, International Nickel Co., and Metal & Thermit Corp. He was a registered professional engineer of New York State.

A touching tribute to Dr. Hess is paid by a colleague, Dr. Matthew A. Hunter, dean emeritus of the R.P.I. faculty, who, since his retirement, had worked with Dr. Hess in the dept. of metallurgy. His statement is an eloquent summation of the admiration of Dr. Hess and the deep sorrow occasioned by his untimely death: "I have been closely associated with Dr. Wendell Hess through every stage of his professional career. His success in his studies in his undergraduate and graduate years had already forecast for him the brilliant career which he displayed in later years.

"He was regarded as an administrator of the highest rank in his work as head of the Department of Metallurgical Engineering and more recently as head of the research activities of the Institute.

"He gained national recognition as the outstanding leader in research in the scientific aspects of welding operations. His all too early death will be mourned by all of his professional associates. In his passing the institute has lost one of its outstanding alumni.

"His monument remains in the hearts of students who were privileged to enjoy an association with him."

Necrology

Date Elected	Name	Date of Death
1928	Colvin B. Brown	Unknown
1913	George O. Deshler	May 13, 1954
1894	Donald G. Forbes	Unknown
1933	Donald C. Gregg	Unknown
1917	A. A. Hoffman	May 4, 1954
1909	Irving S. Josephs	Unknown
1941	Richard Maize	Unknown
1942	Frank E. Mueller	May 21, 1954
1954	Walter Van Rawson	Unknown
1926	Paul T. Seashore	Apr. 11, 1954
1916	Charles B. Strachan	Sept. 9, 1953
1929	William O. Vanderburg	Apr. 16, 1954
1912	C. T. Van Winkle	Mar. 29, 1954

Coming Events

July 16-21, Joint Commission on Electron Microscopy, international conference, Senate House, University of London; London School of Hygiene and Tropical Medicine, Malet St., London, W. C. 1, England.

July 19-20, International Conference, Mechanical Effects of Dislocations in Crystals, University of Birmingham, England.

July 21-28, International Union of Crystallography, general assembly and international congress, Paris.

July 25-Aug. 10, World Power Conference, sectional meeting, Rio de Janeiro, Brazil.

Aug. 2-12, Pan American Federation of Engineering Societies (UPADI), 3rd convention, Sao Paulo, Brazil.

Aug. 23-27, Oak Ridge Summer Symposium, Modern Analytical Chemistry, Oak Ridge, Tenn.

Sept. 8-10, ASME, fall meeting, Schroeder Hotel, Milwaukee.

Sept. 12-16, American Institute of Chemical Engineers, Hotel Colorado, Glenwood Springs, Colo.

Sept. 14-24, Instrument Society of America, Philadelphia.

Sept. 20, AIME, Utah Local Section. Stag. Cocktails, dinner, smoker, Newhouse Hotel.

Sept. 20-24, American Mining Congress, Civic Auditorium, San Francisco.

Sept. 24, AIME, MBD, fall meeting, Fairmont Hotel, San Francisco.

Sept. 28-Oct. 1, Assn. of Iron & Steel Engineers, iron and steel exposition, Cleveland Public Auditorium, Cleveland.

Oct. 1-2, Standards Engineers Society, annual meeting, Haddon Hall, Atlantic City, N. J.

Oct. 3-7, Electrochemical Society, Inc., Statler Hotel, Boston.

Oct. 18-22, National Safety Congress and Exposition, Chicago, Ill.

Oct. 26, Assn. of Consulting Chemists and Chemical Engineers, Inc., annual symposium and banquet, Hotel Belmont Plaza, New York.

Oct. 29, AIME, NOHC and Pittsburgh Local Section, off-the-record meeting, William Penn Hotel, Pittsburgh.

Nov. 1-3, AIME, Institute of Metals Div., fall meeting, Hotel Morrison, Chicago.

Nov. 1-5, American Welding Society, Hotel Sherman, Chicago.

Nov. 1-5, American Society for Metals, National Metal Congress, Palmer House, Chicago.

Dec. 1-4, AIME, Electric Furnace Steel Conference, William Penn Hotel, Pittsburgh.

Dec. 12-15, American Institute of Chemical Engineers, annual meeting, Statler Hotel, New York.

Proposed for Membership

— Metals Branch AIME—

Total AIME membership on May 31, 1954 was 20,762; in addition 1661 Student Associates were enrolled.

ADMISSIONS COMMITTEE

O. B. J. Fraser, Chairman; R. B. Caples, Vice-Chairman; F. A. Ayer, A. C. Brinker, R. H. Dickson, Max Gensamer, Ivan A. Given, Fred W. Hanson, T. D. Jones, Sidney Rolle, J. H. Scaff, John T. Sherman, F. T. Sisco, Frank T. Weems, R. L. Ziegfeld.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

Alabama
Birmingham—Nielsen, Walter N. (M)

California
Berkeley—Wachs, Richard L. (M) (C/S—J-M)

Connecticut
New Haven—Magnus, Donald K. (M)

Delaware
Claymont—Adams, Dwight E. (A)
Claymont—Griffith, Walter G. (A)

District of Columbia
Washington—Reck, David L. (M)

Illinois
Barrington—Harness, Everett (A)
Chicago—Paine, S. Hugh (M)
Chicago—Shapiro, Larry (J) (R. C/S—S-J)

Indiana
Hammond—Engquist, Richard D. (J)
Mishawaka—Lindahl, Frederick A. (M)
New Castle—Atwater, Roy M. (A)

Kentucky
Covington—O'Hara, Eric (A)

Massachusetts
Belmont—Chalmers, Bruce (M)
Springfield—Underhill, George L. (M)

Michigan
Ann Arbor—Carlson, Eugene E. (J)
Ann Arbor—Gallatin, Victor (M) (R. C/S—S-M)
Ann Arbor—Ragone, David Vincent (J)
Dearborn—Scharfenaker, William J. (A)
Detroit—Phelps, Harold R. (M)
Grosse Point Woods—Stoneback, Howard E. (A)

Missouri
Mexico—Beynon, Harold L. (A)
St. Louis—Griffin, Carroll R. (A)

New Jersey
South Orange—Stanwick, Charles A. (M)

New York
Albany—Wilson, Thomas Y. (A)
Buffalo—Campbell, John W. (A)
Loudonville—Reese, Garland W., Jr. (A)
New York City—Baldrey, James H. (M)
New York City—Chapell, Charles H. (A)

Ohio
Cleveland—Ebert, Lynn J. (R. C/S—S-M)
Columbus—Underwood, Ervin E. (M) (C/S—A-M)
Massillon—Williams, Chester B. (A)
Youngstown—Dunkle, Howard C. (M) (R. M)

Oregon
Riddle—Souders, Frank P. (M)

Pennsylvania
Langeloth—Noy, Jack M. (M)
Lebanon—Fegan, Lloyd V., Jr. (J)
Malvern—Thomas, John G. (J)
McKeesport—Sauter, Elroy J. (M)
Pittsburgh—Young, Rush F. (A)
West Mifflin—Matthews, Francis (A)
Wyoming—Gatti, Gerald A. (M)

Texas
El Paso—Armstrong, John W. (J)

Utah
Provo—Aikins, Robert E. (M) (R. C/S—J-M)
Provo—Black, Lawrence F. (M)

Washington
Liberty Lake—Deitz, George A. (J)
Vancouver—Hornbeck, Robert D. (J)

Canada
Sault Ste. Marie, Ont.—Quinn, Randolph B. (M)
Shawinigan Falls, Que.—Desilets, Raoul A. (A)
Sydney, N. S.—McIsaac, Joseph G. (A)
Trail, B. C.—Spencer, Chester M. (M) (R. C/S—J-M)

Chile
Chuquicamata—Towell, Frederick, Jr. (M)

Mexico
Mexico, D. F.—Albarran, Teodoro, Jr. (A)
Mexico, D. F.—McLaughlin, James F. (M)

Sweden
Goteborg—Lundgren, Bengt G. S. (M)

Yugoslavia
Bor, Srbija—Jovanovic, Milan (M)

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* Previous issue.

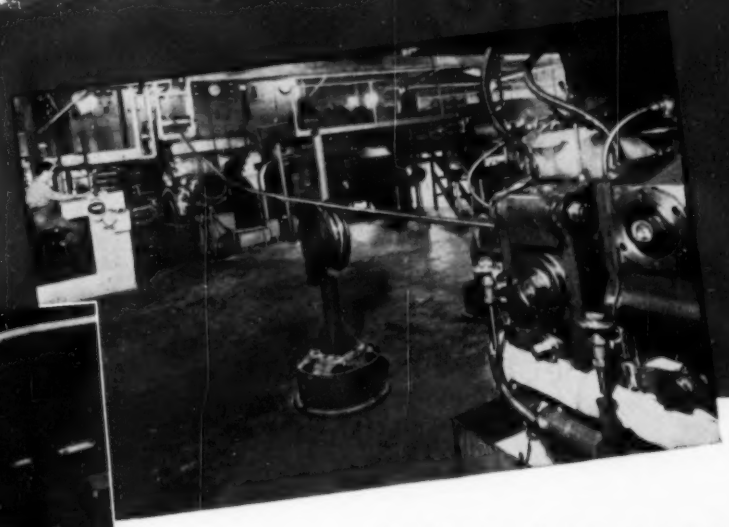
Southwire Company

with the help of

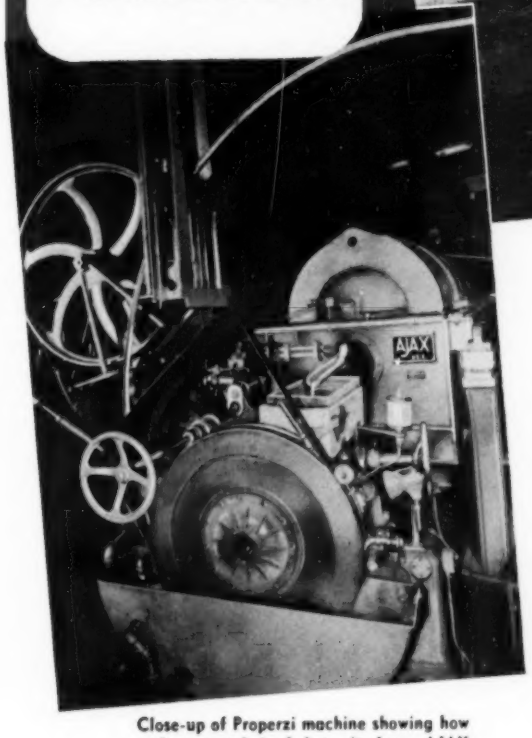
AJAX FURNACES

PRODUCES—

high conductivity aluminum wire direct from the molten metal...



Above photo shows aluminum rod passing from the Properzi casting machine in the background to the rod mill in the foreground. AJAX furnace is at right of the Properzi machine.



Close-up of Properzi machine showing how molten metal is fed to it from AJAX furnace.

In connection with the well-known Properzi method, the Southwire Company of Carrollton, Georgia, is using an AJAX low-frequency induction furnace for the quick and efficient melting of high purity aluminum.

This combined melting and holding furnace is ideally suited to mass or continuous production processes, because the molten metal can be fed in a steady stream from the holding reservoir of the furnace, which in turn is kept supplied with metal from the melting reservoir.

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Write for Reprint of Article Entitled "ALUMINUM: Continuous Casting Gaining"

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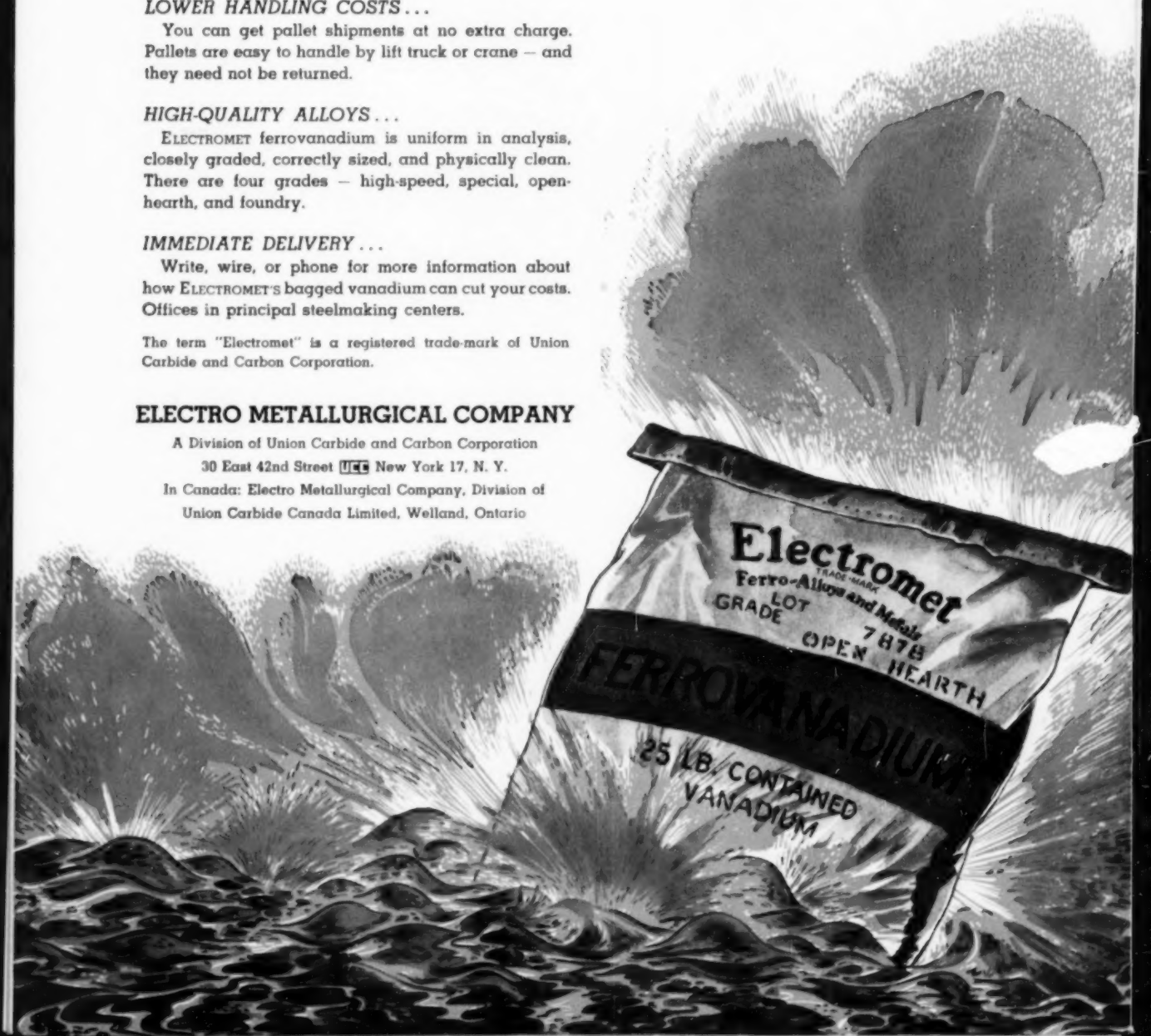
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